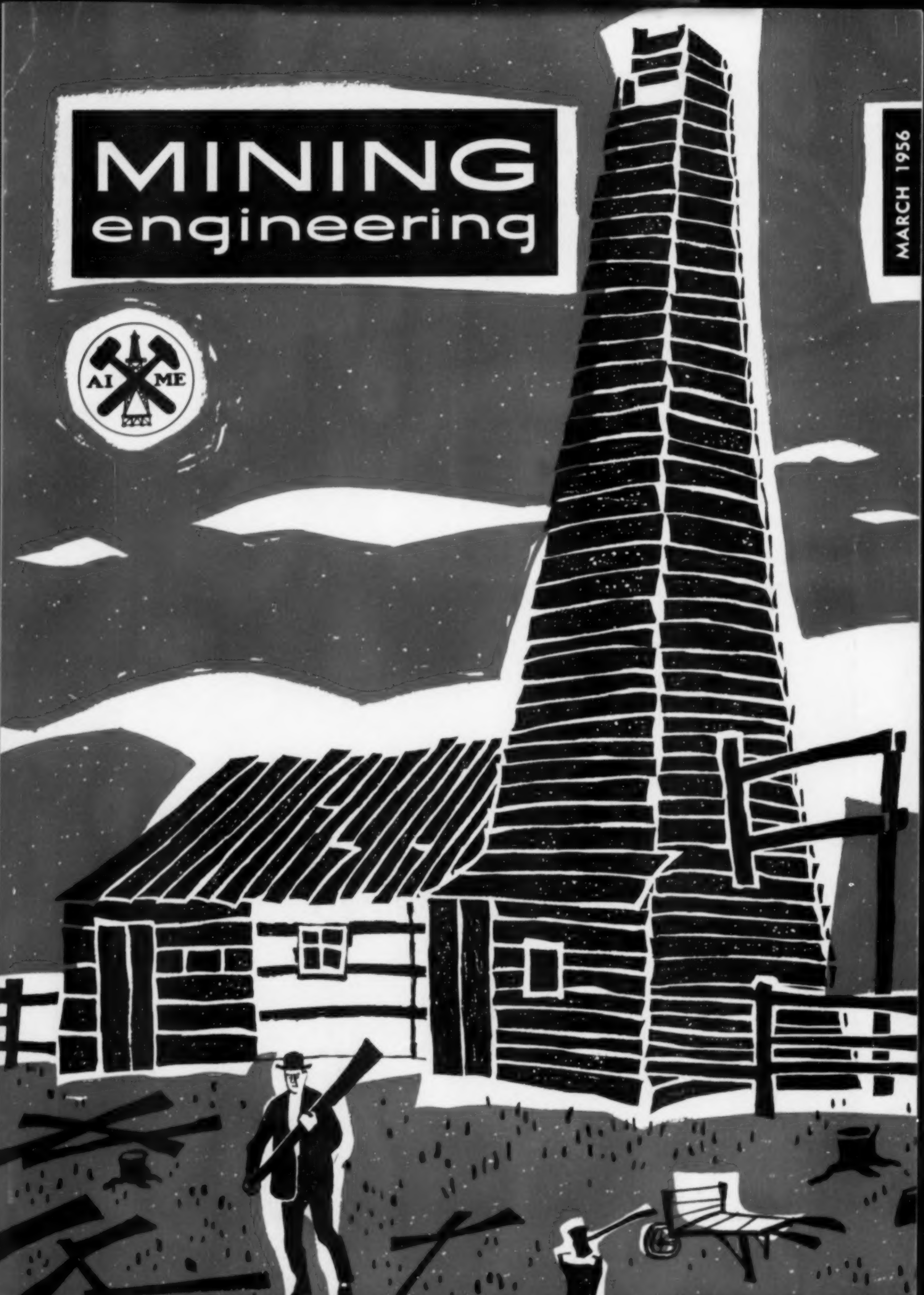
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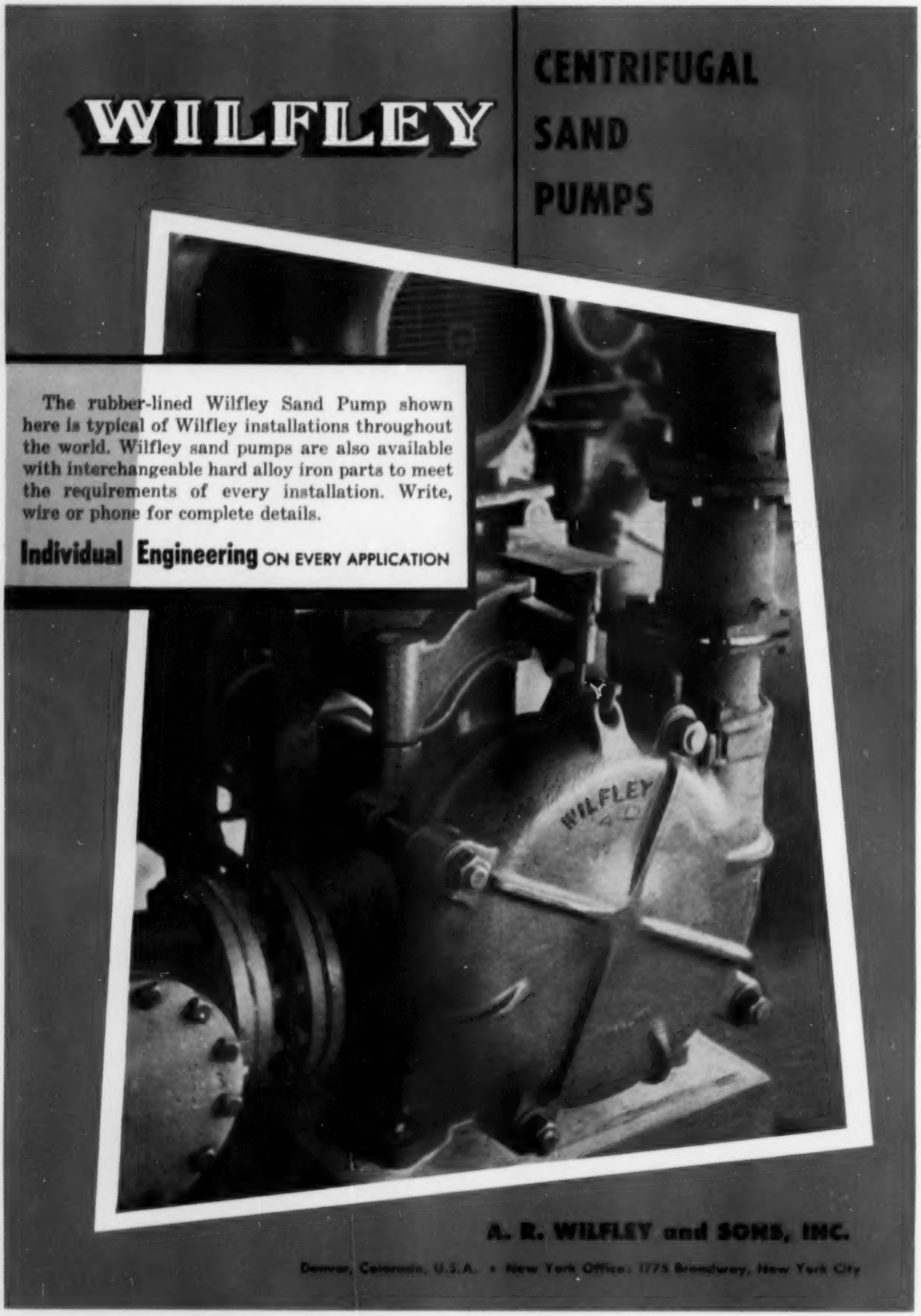
WILFLEY
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PUMPS
The rubber-lined Wilfley Sand Pump shown here is typical of Wilfley installations throughout the world. Wilfley sand pumps are also available with interchangeable hard alloy iron parts to meet the requirements of every installation. Write, wire or phone for complete details.
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MINING engineering

VOL. 8 NO. 3 MARCH 1956

COVER

The Mining Branch salutes the recognition of the stature of the Petroleum Branch of AIME in the new seal—and reminds its readers of the petroleum industry's early days in the sketch of Colonel Drake's original well at Bradford, Pa.—the well that started a river of black gold in 1856, only seven years after the discovery of gold in California.

FEATURES

ARTICLES

TRANSACTIONS

PERSONNEL

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Mining Engineer—Public Relations, 26, married. Experience: technical magazine writing, publicity, industrial relations, economics. Mining training, research experience: metallic and coal mining. Colorado School of Mines, Boston University graduate, B.S. public relations, M.S. mining engineering. Desires position public or industrial relations, management consulting or related fields with rapidly growing mineral industry. Available July. M-255.

Civil—Mining Engineer, U. S. citizen, 26, married, one child, B.S. in civil engineering. Six years varied experience underground hardrock mining, triangulation, Corps of Engineers construction work, industrial plant operations. Desires responsible position mining or construction, western U. S. or foreign. Employed but available short notice. M-257-13-SF.

Exploration—Economic Geologist, B.Sc., M.Sc., 32, excellent health; nine years exploration experience North and South America. Desires responsible position abroad. Excellent references. M-256.

Mining Geologist, Mine Engineer, 34, married. Seven years experience exploration geologist; more than 15 years diversified engineering experience. Speaks Spanish. Prefers foreign location. M-259-14-SF.

Metallurgical Engineer, 29, married. Eighteen months assistant preparation engineer with large anthracite producing company, 28 months instructor in mining extension with large eastern university, edited text on coal preparation, one year nonmetallic mining and beneficiation. B.S. and M.S. in mineral dressing. Currently employed. M-258.

POSITIONS OPEN

Assistant Mine Superintendent for metal mine. Must have had 8 to 10 years underground mine operating experience preferably with Latin Americans. Working Mexican passport must be obtained for applicant and family. Must be in good physical condition. Salary, $7200 per year, U. S. currency, plus bonus, low income tax, and free house for family. Location, Mexico. S-238.

Engineers. (a) General Superintendent, 35 to 45, mining or metallurgical engineering graduate, with at least five years supervisory experience in metal mining and milling. Salary, $8,000 to $10,000. (b) Staff Engineer, young, mining or civil engineering graduate, to assist chief engineer covering open pit mine surveys, designing and installation of ore handling and milling equipment. Salary, $4800 to $6600 a year. Location, South. W2850.

Assistant General Superintendent, 35 to 45, mining engineering graduate, with supervisory production experience in underground mine and operation of mill equipment. Salary, $8,000 to $10,000 a year. Location, East. W2770.

Research Director, Ph. D., required for large industrial research laboratory specializing in metal fabrication. Salary, $25,000 to $50,000 a year. Location, Mid-Atlantic States. W2855.

Mining Engineer, 30 to 40, with five to ten years experience in layout work for development and exploration on stope nonmetallic mining. Salary open. Location, upstate New York. W2748.

Mining Engineer, preferably single, or married man on single status basis. Experienced in mine surveying, sampling, mapping, and field engineering and surveying. The operation is 160 to 200 tons per day silver-lead-zinc mining in limestone replacement deposit. Salary, $5100 to $5400 a year, plus room, board, and transportation. Location, Central America. F3007.

GRADUATE ENGINEER — Position as superintendent of rotary kiln operations. A minimum of five years experience as superintendent or assistant superintendent. Submit resume of education, work, personal history.
Box 2-A AIME
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SALES ENGINEERS
Explosives manufacturer with nation-wide distribution has attractive openings for men with technical background or practical experience as sales engineers; age, under 30 preferred. Starting salary commensurate with experience. Considerable traveling is required. Liberal benefits. Every incentive for advancement.
This is an excellent opportunity for the application of your knowledge of mining, quarrying, construction, and related industries.
Send complete resume. Photograph optional. All correspondence will be held in strict confidence, and references will not be contacted without permission of applicant.
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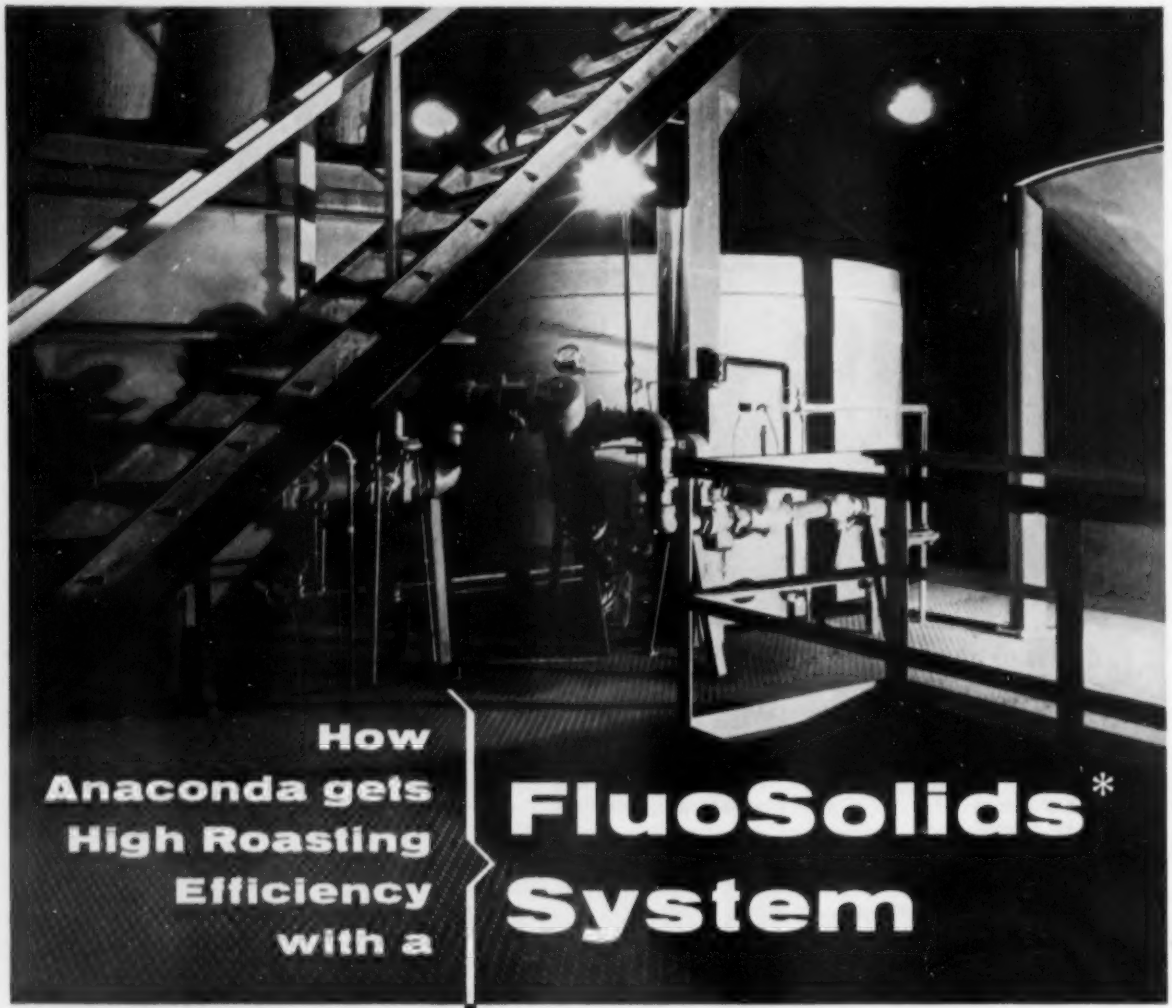

For sponge iron production at Anaconda, Montana, Anaconda Company needed a calcine containing less than 1% sulfur. Their conventional fixed bed pyrite roasters, producing gas for an acid plant, delivered a calcine averaging 2 to 6% sulfur. The problem was solved by installing a Dorrco FluoSolids System for an additional roasting stage.

Unique in this installation is the fact that roasting is carried out autogenously on pyrite containing as little as 2% sulfur. Only outside fuel required is for starting up the System.

What's more, with sulfur in the feed all the way from 1 to 9%, calcine from the FluoSolids System consistently contains 0.8 to 0.9% S. The FluoSolids Reactor has an inside diameter of 10′ and handles 200 TPD at 1200°F.

High roasting efficiency is just one of the many advantages of fluidization. If you'd like more information on the Dorrco FluoSolids System, the most significant advance in roasting techniques in the last 30 years, just drop a line to Dorr-Oliver Incorporated, Stamford, Connecticut.

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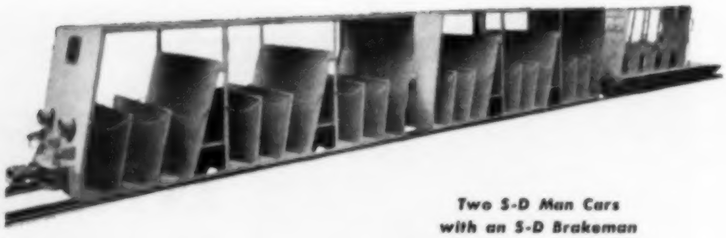
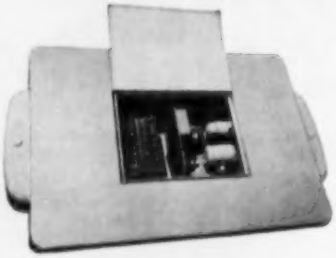
An S-D Granby



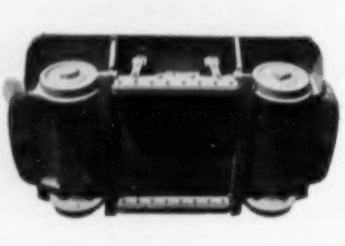
Two S-D Man Cars
with an S-D Brakeman

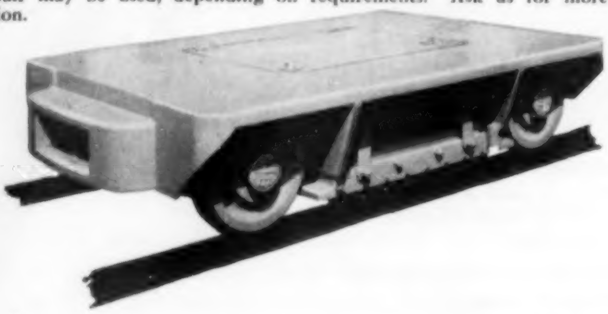
Sanford-Day ... Super

Now you can eliminate skids and additional locomotives for braking trips downgrade!

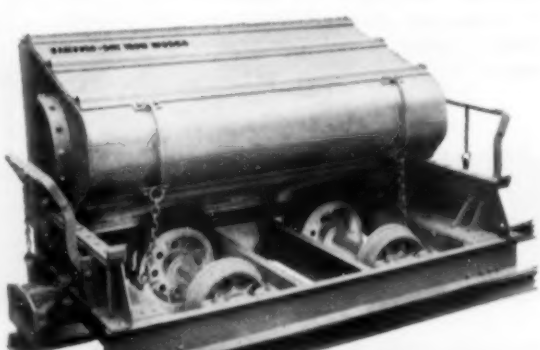


YOU NEED NO LONGER resort to skids or costly locomotives for extra braking power. We have developed a special car for braking that incorporates electrically operated magnetic brake shoes. Several are already on the job in metal mines. The S-D Brakeman Car is built to meet your particular requirements. It may be used between locomotives . . . between locomotive and cars . . . between cars or at end of a trip to prevent runaways due to coupler failures, etc. Controls can be provided at locomotive operator's fingertips. Top photo at left shows the compartment for batteries and motor-generator set. Lower photo shows the pair of brake shoes. One or several pair may be used, depending on requirements. Ask us for more information.





An
S-D
Rocker



An S-D Heavy-Duty Rocker
with outside frame construction

BOOKS

Please Order These Publications from the Publishers

Handbuch der Mikroskopie in der Technik, edited by Hugo Freund, *Umschau Verlag*, Frankfurt. Information and literature on this eight-volume work on miscroscopy in technological fields is available from E. Leitz Inc., 468 4th Ave., New York 16, N. Y. Volumes may be bought as a set or individually. They are from $30.00 to $40.00 each and only in German.

First Annual Bituminous Conference, *University of Minnesota*, Center for Continuation Study, Minneapolis 14, Minn., free, various pagings.—This conference, presented in cooperation with the Minnesota Bituminous Pavement Assn. and the Asphalt Institute, took place Nov. 3, 1954.

Tungsten, The Story of an Indispensable Metal, by Mildred Gwin Andrews, *The Tungsten Institute*, 1757 K St., N.W., Washington 6, D. C., $1.50, 28 pp., 1955.—An interesting and informative digest prepared for the general public, schools, colleges, and the quick reference shelf.

Coal Mine Modernization, 1955, *American Mining Congress*, Ring Bldg., Washington 6, D. C., $3.50, 410 pp., $3.50.—This volume, containing the complete proceedings of the 1955 American Congress Meeting on this subject, is made up of nearly 50 papers, with discussion, presented by representatives of operators and machinery and equipment manufacturers. Subjects dealt with include continuous mining methods, mechanical mining, roof supports, haulage, power, strip mining, auger mining, coal preparation, maintenance, management, and safety.

Geology and Ground-Water Resources of Graham County, Kansas, by Glenn C. Prescott, Jr., Bulletin 110, 98 pp, 7 fig., 10 pl., September 1955. Mailing charge 25¢. **Petrography of Upper Permian Rocks in South-Central Kansas,** by Ada Swineford, Bulletin 111, 179 pp., 13 fig., 24 pl., May 1955. **Stone Corral Structure as an Indicator of Pennsylvania Structure in Central and Western Kansas,** by Daniel F. Merriam, Bulletin 114, part 4, 24 pp., 11 fig, June 1955. Mailing charge 10¢. These bulletins are available from the *State Geological Survey of Kansas*, University of Kansas, Lawrence, Kans.

Minerals Yearbook, Metals and Minerals (Except Fuels), 1952, Volume I, by the staff of the U. S. Bureau of Mines, Superintendent of Documents, *U. S. Government Printing Office*, Washington 25, D. C., $4.00,

(Continued on page 248)

BOOKS

(Continued from page 244)

1218 pp., 1955.—This volume is made up of chapters on mineral commodities, both metals and nonmetals, except mineral fuels. There is also a chapter reviewing these mineral industries, a statistical summary and recapitulation, and chapters on mining technology, metallurgical technology, and employment and injuries. In 1952 the national income derived from metal and nonmetallic mining and quarrying industries was 4 pct above the 1951 level. The United Nations' world mining index (fuels and nonfuels) average for 1952 was 3 pct greater than for 1951.

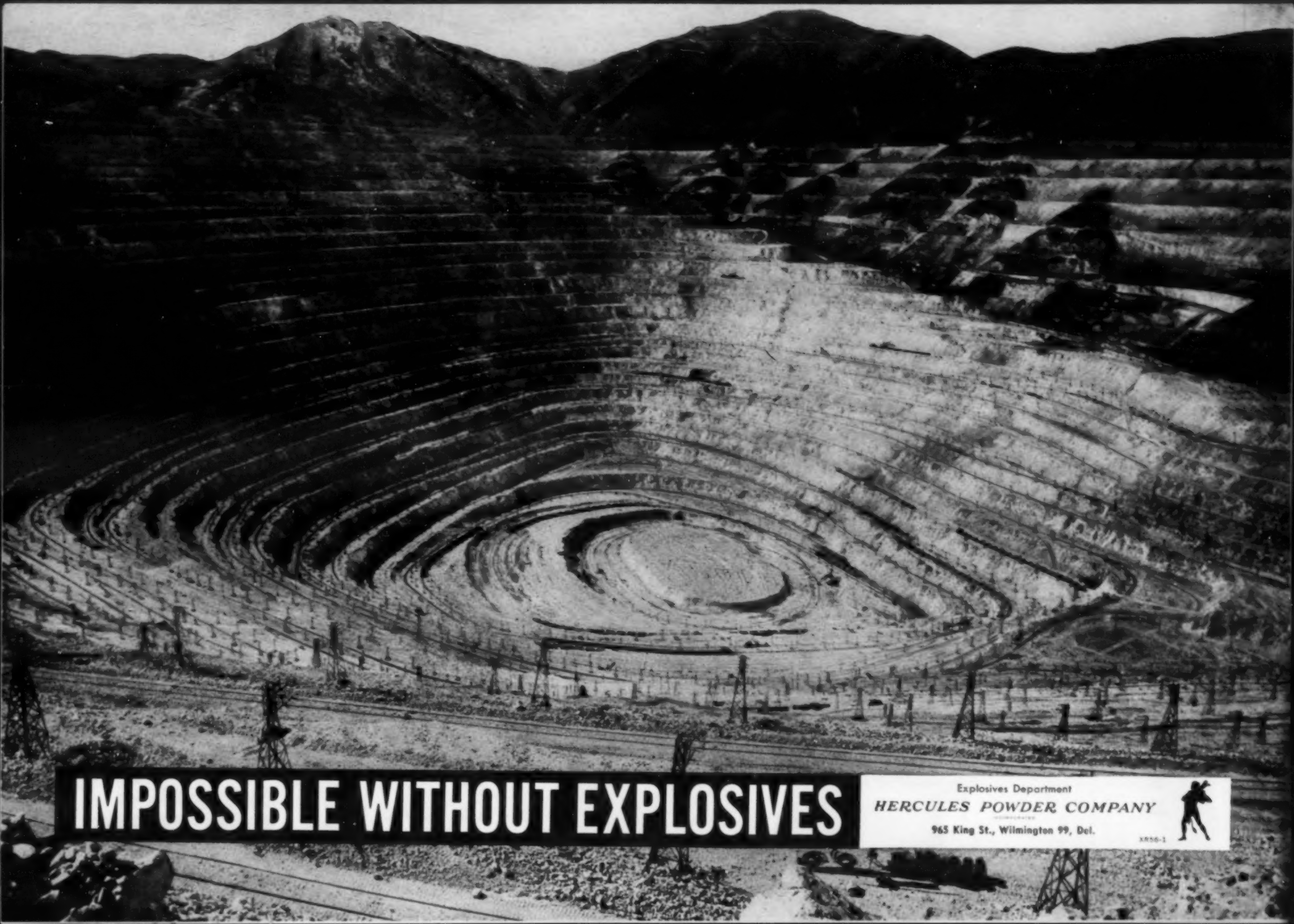
Crane Handbook, *Whiting Corp.,* 157th and Lathrop, Harvey, Ill., 176 pp., 1956.—Practical data on electric traveling cranes. Seventeen sections include: terminology and classification; operating speeds; overhead clearances; design; safety features; operation and maintenance instruction; auxiliary equipment; modernization of old cranes. Available free of charge to qualified engineers and others directly interested in crane specification. Make application on business letterhead.

Silica Resources of Clark County, Nevada, by Thomas D. Murphy, *Nevada Bureau of Mines,* University of Nevada, Reno, Nev., Bulletin 55, 50¢, 33 pp., 1954.—Prepared cooperatively with the U. S. Geological Survey. Illustrated.

California Journal of Mines and Geology, Volume 51, No. 4, October 1955, *California Division of Mines,* Ferry Bldg., San Francisco 11, Calif., $1.00, 465 pp., map in pocket.—Contents include an article by Henry H. Symons and Fenelon F. Davis on California mineral commodities in 1952 and 1953 and an article by Mr. Davis on mines and mineral resources of San Mateo County.

Steels for the User, R. T. Rolfe, *Philosophical Library,* $10.00, 399 pp., 3rd ed., 1956.—This book was written "to bridge the gap between science and practice for carbon steels in industry." Although it deals substantially "with carbon steels, alloy steels are discussed for duties for which carbon steels are unsuitable, as with nitriding steels and those for high temperature service." The author is consultant to the British firm, W. H. Allen Sons & Co. Ltd. This third edition has been enlarged and brought up to date.

The Pittsburgh No. 8 and Redstone No. 8A Coal Beds in Ohio, by Richard M. DeLong, *Ohio Div. of Geological Survey,* Orton Hall, Ohio State University, Columbus 10, Ohio, RI 26, 50¢, 49 pp., 27 tables, 12 figs., 1955.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Manufacturers News

News
Equipment
Catalogs

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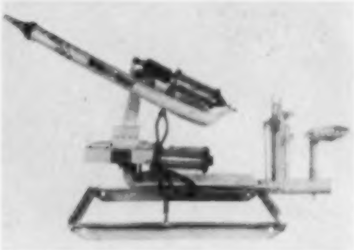
Hydraulic Monitor

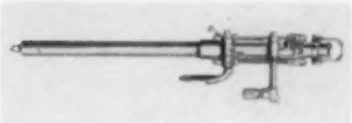
Recent addition to the *Chiksan* Intelli-Giant line is the 650-lb model shown below. The 4-in. monitor throws a 300-ft stream through a



2½-in. nozzle tip at the rate of 2000 gpm. One-man operated at gun or remote station, the MH-4 has a horizontal sweep of 270° and a vertical range of 120°. **Circle No. 1.**

Lightweight Stoper

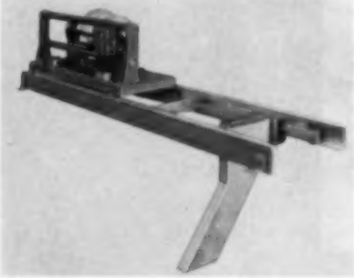
Le Roi Div., *Westinghouse Air Brake Co.*, now makes a fast lightweight stoper for metal mining that is said to reduce operator fatigue. Weighing only 79 lb when used with



a 26-in. feed, the S-10 operates on 70 to 90 lb of air pressure and is available with 26, 32, or 38-in. feeds. Diameter of air hose is ¾ in., water hose ½ in. **Circle No. 2.**

Automatic Sampler

Denver Eqpt. Co. has an automatic sampler with an end carriage that provides accurate sampling results in normally inaccessible places. By taking the whole of the stream part



of the time, sampler eliminates errors resulting from variations across the stream. Intermittent operation is controlled by a Telechron time switch. **Circle No. 3.**

Magnetic Separator

For purification of silica sand, borax, feldspar, and many other dry granular products, *Exolon Co.* has two laboratory model high intensity magnetic separators. These test separators consist of a rotor mounted on extra heavy ball bearing pillows, driven by a totally enclosed ball bearing motor through a variable speed drive adjustable from approximately 80 to 300 rpm. **Circle No. 4.**

Rock Drill

The *Thor Power Tool Co.* 77 rock drill has a recently developed latch-type front head. This assures a positive lock when pulling drill steels, if mounted on a sinker leg. Unit is air-operated, weighs 55 lb, and drills holes up to 20 ft in hard rock. It is ruggedly constructed for shaft sinking, quarrying, and road building. **Circle No. 5.**

Magnetic Susceptibility Meter

The *Mt. Sopris* magnetic susceptibility meter may be used for field or laboratory measurements. Instrument is basically a mutual inductance bridge that employs a three-coil sensing element for minimizing errors caused by irregular rock surfaces. Narrow band amplification reduces power line pick-up to a negligible point with both field and laboratory coils. **Circle No. 6.**

Push-Button Man Cage

Connellsville Mfg. & Mine Supply Co. has a custom-built elevator-type man cage that is fully automatic. Special double-deck steel cage is built to operate in a circular shaft. New-type core drill is said to sink this shaft quickly and economically. All parts can be moved by truck. Cage, hoist, and auxiliary equipment can be dismantled and reassembled in about ten days. **Circle No. 7.**

Tubeless Truck Tire

B. F. Goodrich Tire & Equipment Div. has a tubeless truck tire with a special X-99 tread compound that helps prevent tread cracking and checking. When mounted on new drop center rim assembly, it can mean an overall weight reduction of about 40 lb per wheel. The Power Express tubeless also runs cooler than conventional tire and tube. **Circle No. 8.**

Radon Gas

Atomic Engineering Corp. has a radiation counter that measures radon gas concentrations in mine shafts, drifts, and tunnels. Known as the Utah Juno, this radon gas sur-

vey unit is highly sensitive to alpha, beta, and gamma radiation. It weighs 8 lb and radiation concentrations are shown on a large, easy-to read meter. **Circle No. 9.**

High-Speed Vibrating Screen

Hewitt-Robins has a high speed vibrating screen claimed to be 30 to 80 pct faster than previous mechanical screens. The HS Vibrex has a speed of 3300 rpm when set for 1/32-in. stroke and 2400 rpm on a 1/16- in. stroke. Atomized oil system prevents overheating of bearings formerly encountered with conventional grease and lubricating oils. It has been developed for the fine screening of sand, clay, fertilizers, chemicals, coke, coal, and asbestos. **Circle No. 10.**

Tough Hose

Manhattan Rubber Div., *Raybestos-Manhattan Inc.* has a strong, lightweight hose built for pressures up to 2000 psi. Special steel braids offer resistance to accidental crushing in mine and quarry service. Neoprene tube withstands hot oil from compressors and bright yellow cut-resistant cover provides maximum visibility and hose life. **Circle No. 11.**

Rear Dump Trailer

PR 15-Cat DW 15 rear dump trailer, made by *Athey Products Corp.*, has a 15.6-cu yd capacity. PR 15 has speeds up to 31.3 mph, right or left angle turns of 90°, and three-stage hydraulic hoists that tilt the body 60°. Loaded unit weighs 84,683 lb with 37 pct of load on big drive wheels of the Cat 186-hp tractor. **Circle No. 12.**

New Thor all-purpose one-man drilling machine

THOR introduces the powerful new 390 drilling machine with a host of features which speed up drilling operations and reduce operator fatigue.

This Thor drill is equipped with heavy-duty aluminum cylinder and chrome-plated steel piston rod and a specially designed 45 lb. sinker rock drill with integral feed leg connection. It can be easily converted for sinking. Thor 390 is equipped with a line oiler and automatic water valve which can be replaced with a plug for dry operation or collaring.

All controls are grouped on the backhead for safe convenient operation. A six position throttle, a Thor exclusive, gives a complete range of control from complete shut-off of air and water, to full air and water supply in logical steps for easy convenience and instant control.

Optional feed travel legs available in 36", 48" and the new model 680 72" telescopic leg.

Your nearest Thor representative will be glad to arrange for a demonstration. Thor Power Tool Co., Aurora, Ill.

Thor
PORTABLE POWER
TOOLS

New Thor 390—drifter, sinker, stoper—all in one

The Thor No. 390 is easily adaptable for all types of drilling at all angles. The drill can be removed from the leg for use as a sinker by loosening only one nut. Tool swivels into vertical position for stoping.

THOR POWER TOOL COMPANY

Atlanta, Birmingham, Boston, Buffalo, Chicago, Cincinnati, Cleveland, Denver, Detroit, Houston, Los Angeles, Milwaukee, Newark, Long Island City, N.Y., Philadelphia, Pittsburgh, St. Louis, San Francisco, Seattle, Toronto, Canada, Export Division, New York City

(21) SPECIAL STEELS: *Allegheny Ludlum Steel Corp.* has a 16-page booklet on stainless steels, electrical materials, Carmet carbide materials, and tool steels. Included are charts analyzing various steels, as well as information on fabrication and properties.

(22) SOLID FILM LUBRICANTS: B-3 brochure from *Electrofilm Inc.* shows the technical advances of solid film lubrication. It is claimed that this process inhibits fretting corrosion, prevents galling and seizing, lubricates under high loads and speeds from −100°F to +700°F (in some applications to 1100°F), and always produces dry, clean, and dust-free operation.

(23) SHOCK-CUSHION: CR-403-F from *International Harvester Co.* explains the Hydro-Spring on the International Drott Skid-Shovel. Hydro-Spring is said to reduce effects of shock forces on tractor, loader, and operator, and to extend shovel-tractor life by 25 pct.

(24) EMERGENCY CHART: *Fisher Scientific Co.* has an 18x27-in. laboratory chart listing emergency treatments for poisons, accidents, and corrosive chemical burns. Treatments are up to date, for example, salves and ointments are no longer recommended for phenol burns. Illustrated is the back pressure-arm lift method of artificial respiration advocated by the Red Cross and Civil Defense agencies.

(25) DATA CATALOG: Available from *Lefax Publishers* is a catalog of pocket size technical data books. A few of the many subjects are: reinforced concrete, metallurgy, mining geology, mining engineering, and conversion tables. Books sell for $1.25 each.

(26) HYDRAULIC SYSTEMS: Industrial products dept., *Sun Oil Co.*, has a "Trouble Shooting Chart for Hydraulic Systems," designed to help operators and maintenance men spot and cure common troubles.

Free Literature

(27) INSTRUMENTATION: *Illinois Testing Laboratories Inc.* has a general bulletin listing salient features of its pyrometers, thermometers, thermocouples, and other instruments.

(28) MOTOR SCRAPER: *Allis-Chalmers'* catalog MS 663 presents features of the TS-360 and outlines its operator comforts. This motor scraper has a capacity of 20 cu yd heaped or 15 cu yd struck. It is powered by a 280-hp diesel engine.

(29) NO SCRAP: Bulletin 301 from continuous-cast products dept., *American Smelting & Refining Co.*, lists all stock sizes and weights of

solid and hollow bronze bars from ½ to 9-in. diam. Asarcon 773 is claimed to save money because of length, size, machining time, metallurgical characteristics, and physical properties.

(30) VALVES: Made by *Grinnell-Saunders*, the Straightway diaphram valve is used for handling sludges, viscous materials, fibrous slurries, and corrosive chemicals. Advantages include elimination of flow restrictions and minimum pressure drop.

(31) DISC ROLL MILL: The *Hardinge* Disc Roll mill is a roller-type mill based on the well known German Loesche mill. U. S. improvements include the Gyrotor classifier and pneumatic roll pressure control. It is particularly suited for grinding relatively soft minerals, such as limestone, coal, phosphate rock, bauxite, and the like.

(32) H-F COMBUSTION UNIT: Bulletin LEP-1 from Laboratory Eqpt. Div., *Lindberg Engineering Co.*, is on a high frequency combustion unit. It is used for rapid determination of total sulfur in diesel fuel, kerosene, oil sludges, lubricating oils, and other inorganic materials.

(33) NUTS & BOLTS: Quality track hardware is necessary to prevent track failure and down time. *Caterpillar's* "Let's Talk Nuts and Bolts" shows the importance of maximum strength and toughness, high standards, and careful inspection.

(34) FREIGHT CAR HANDLER: Made by *Whiting Corp.*, the Trackmobile hauls, switches, and spots as many as six loaded railroad cars at once. Machine travels sideways on rail wheels. With its pneumatic road wheels lowered, it moves onto the next track or doubles as a tractor for pulling carts and skids.

(35) BRONZES: "Engineering Properties and Applications of Ni-Vee Bronzes" is available from *International Nickel Co.* Booklet recommends Ni-Vee types for constructional bearing and pressure castings and explains cross-over applications. High "as cast" properties can be readily improved by simple heat treatment.

MAIL THIS CARD

for more information on items described in Manufacturers News and for bulletins and catalogs listed in the Free Literature section.

3

Mining Engineering **29 West 39th St.** **New York 18, N. Y.**

Not good after June 15, 1956—if mailed in U. S. or Canada.

Please send { More Information ☐ / Price Data ☐ / Free Literature ☐ } on items circled.

Name____________________ Title________

Company____________________

Street____________________

City and Zone____________________ State________

1	2	3	4	5	6	7	8	9	10
11	12	13	14	15	16	17	18	19	20
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41	42	43	44	45	46	47	48	49	50
51	52	53	54	55	56	57	58	59	60
61	62	63	64	Students should write direct to manufacturer.					

(36) FLOTATION: Want fast, complete separation of sulfide minerals at minimum cost? Information from *Dow Chemical Co.* will show you how Dow Xanthates are improving metallurgy and saving dollars "in mills all over the world."

(37) ALUMINUM BUILDINGS: *Alcoa* has five booklets on aluminum architectural and industrial building products. Included are specification suggestions for using colored aluminum and a discussion of the unique industrial sandwich wall system.

(38) LIFT TRUCK: Leaflet from *Hyster Co.* contains specifications on a gasoline-powered lift truck. The TC-200 has a capacity of 20,000 lb at 24-in. load centers and a speed of more than 20 mph in either forward or reverse.

(39) CRANE-EXCAVATOR: *Schield Bantam Co.* has a 4-page bulletin CR-501 on a ⅜-cu yd, 6-ton self-propelled crane-excavator. Model features one-man operation, two-speed independent travel, a no-shift forward-reverse travel design, and a 19½-ft outside turning radius.

(40) ELECTRIC MOTORS: Bulletin 1700 from *Louis Allis Co.* includes information on open drip-proof as well as enclosed and explosion-proof electric motors.

(41) AUSTRIAN COMPRESSOR: *Warsop Power Tools Ltd.* is distributor for the Jenbacher Werke diesel-driven air compressor. This two-wheeled trailer unit is completely self-contained. Speed can be varied between 800 and 1500 rpm.

(42) POWER UNIT: Bulletin MS-455 from *Allis-Chalmers* gives the design, engineering, and performance story of the 4-cyl, 60-hp W-226 power unit. Included is a list of special and extra equipment to increase the unit's versatility.

(43) COAL DUST CONTROL: Bulletin 551-D from *Wheelabrator Corp.* on the control of dust from coal handling operations contains four illustrated case histories. Discussed are local exhaust ventilation and cloth-tube-type dust collectors.

(44) SEISMOGRAPH: *W. F. Sprengnether Instrument Co.* has a bulletin on a portable seismograph recording blast vibrations. This three component instrument is self-contained in a rigid aluminum case, 25x10x8 in., weighs only 38 lb, and requires no external connections.

(45) WIRE ROPE: *E. H. Edwards Co.* has a 32-page brochure with detailed information on splicing and fitting wire rope. Step-by-step photographs and drawings illustrate correct method of making various splices, breaking down a strand, blocking, and serving. Also included are sections on eight strand ropes, grommets, and attaching a socket.

(46) TRANSPORTATION: Catalog 2 shows a few of the many different types of standard and special transportation equipment, made by *Easton Car & Construction Co.* Pictured are off-highway trailers, truck bodies, mine and quarry cars, Cross-Bay motor driven transfer cars, intraplant trailers, and both light and heavy duty industrial cars.

(47) TORQUE CONVERTERS: *Twin Disc Clutch Co.* publishes a quarterly, "Production Road." Among other articles, latest issue contains "How Torque Converters on Shovels Increase Efficiency." It is illustrated with charts and on-the-job photographs.

(48) PROCESS PLANTS: Bulletin 2514 from Chemical Plants Div., *Blaw-Knox Co.*, outlines design, engineering, construction, and initial operation of process industries plants. Among industries covered are nuclear energy, metals processing and treatment, and petroleum and petrochemicals.

(49) LABORATORY OVENS: Bulletin 5520 from *Soiltest Inc.* deals with Blue M electric ovens, furnaces, and related equipment. Given are dimensions, temperature ranges, and prices.

(50) HEAVY-MEDIA: In *Wilmot's* HM vessel the entire separating process is performed with only the mechanism and power required for lifting the sink from pool. Other advantages include minimum floor space, few moving parts, and continuous visual inspection.

(51) TUNNEL DRIVING: Bulletin L-1005C is on low cost, high speed tunnel driving. Photographs, drawings, and statistics tell the story of big jobs done with *Eimco* equipment. Among them are the Duchesne water tunnel, the Carlton tunnel, the Gateway tunnel, and the Continental Divide tunnel. The last is the longest tunnel ever to be driven from two headings in the U. S. with no intermediate openings.

(52) TWO-STAGE PUMPS: *Allis-Chalmers'* bulletin 52B6105C is on two-stage pumps, which are available in close-coupled and frame-type construction. Capacities are from 300 gpm at heads of from 300 to 550 ft at temperatures to 250°F. Pumps are suitable for boiler feed, mine service, humidifier, and other high pressure applications. They are also used for gathering, loading, and pipeline operations in oil fields.

(53) FLOTATION AGENT: Want maximum recovery? *Hercules'* rosin amine D is a "highly efficient collector" in the flotation of phosphate feldspar and mica. Information is also available on Terposol 50 and other flotation agents. Terposol 50 has displayed "excellent frothing properties in the flotation of sulfide and nonsulfide minerals."

(54) LIFTS: *Oster Mfg. Co.* has an 8-page brochure on multipurpose, manually propelled, hand and battery-powered hydraulic lifts with capacities from 500 to 2000 lb. Design provides four carriers in one: platform truck, straddle fork truck, portable elevator, and shop crane.

(55) FLUID DRIVE: *American Blower Corp.* has a catalog on the Gyrol fluid drive type VS class 2F, a packaged self-contained unit for power transmission and control. It is designed for oil rig application; transmitting power from the engines to the rig's draw works, rotary, and slush pumps.

(56) SHAFT EQUIPMENT: Shown in bulletin 20 from *Mayo Tunnel & Mine Equipment* are head frames, skips, cages, gilleys, and jumbos. On-the-spot photographs show deep and shallow shafts and other units. Also illustrated is the Mayo self-dumping gilley designed for small shafts.

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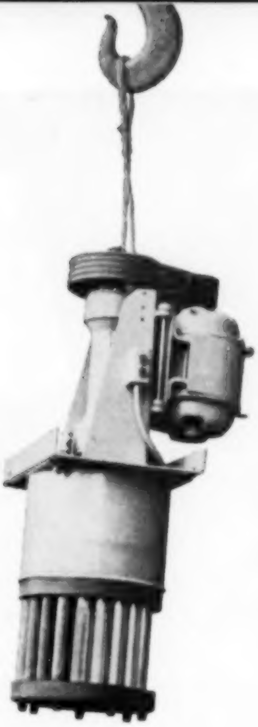
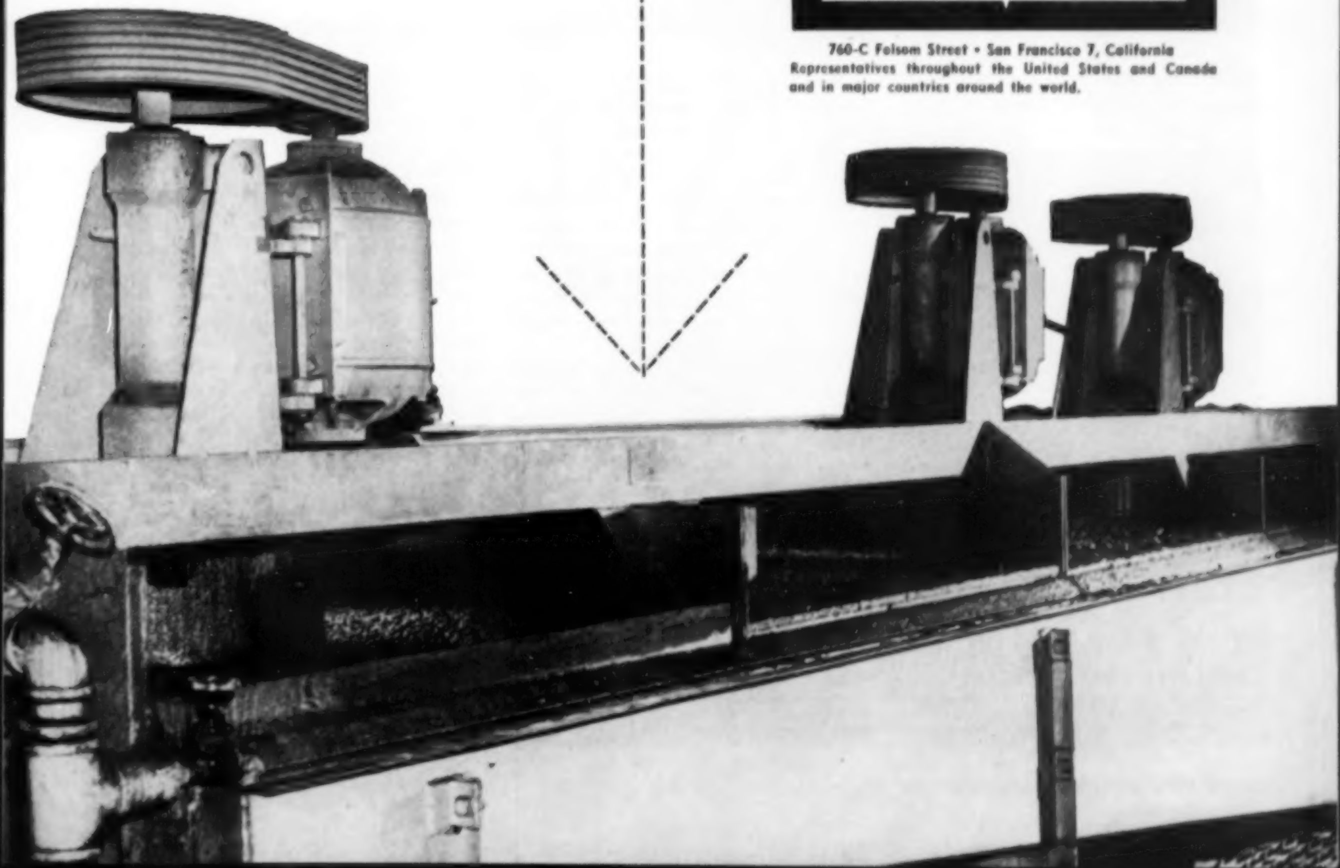
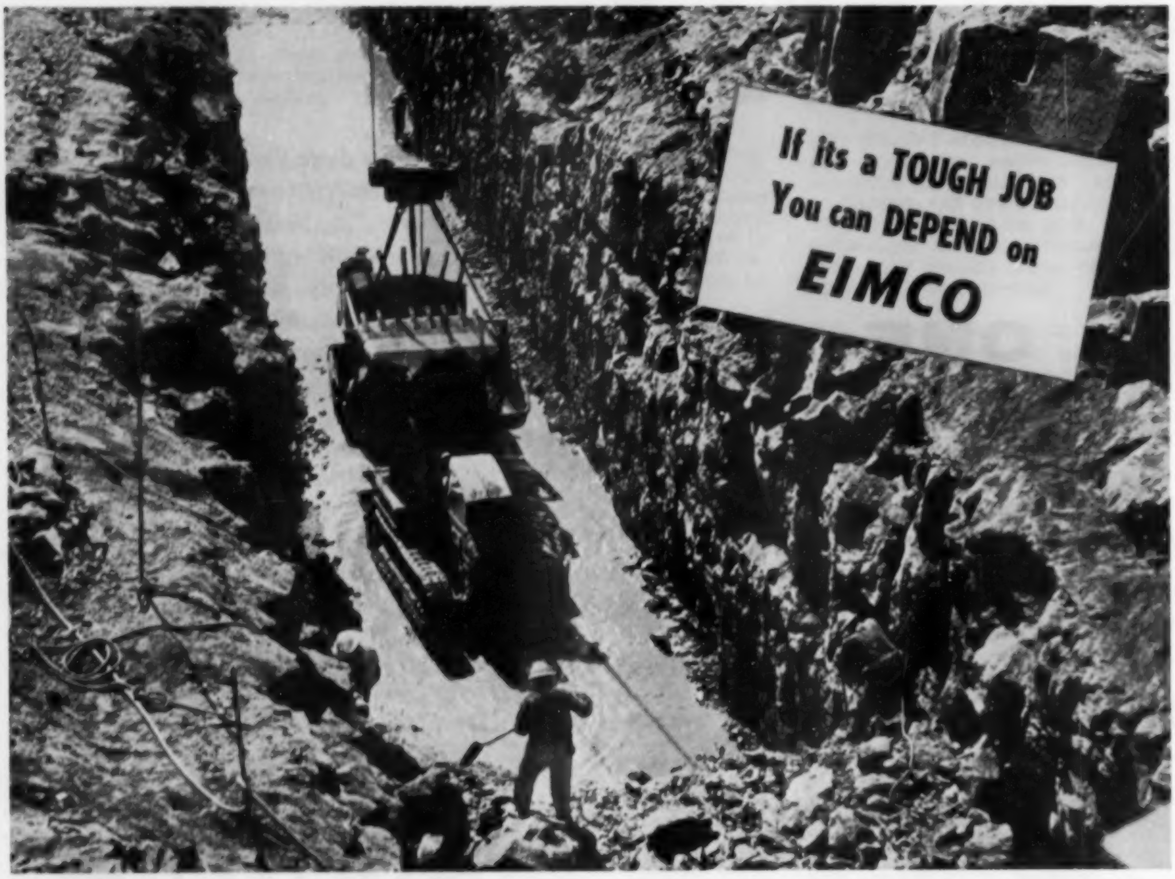




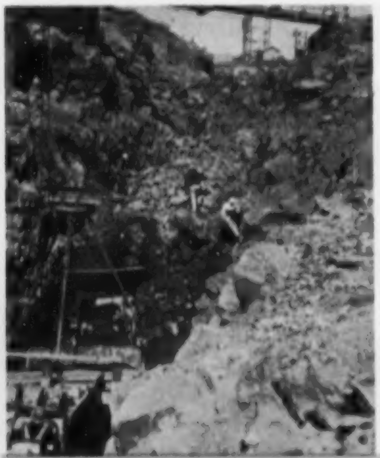


EIMCO KEEPS TOUGH EXCAVATION JOB ON SCHEDULE

Contractor — Montag - Halvorson - Cascade-Austin.

Location — The Dalles, Oregon.

Project — The Dalles Dam — Excavation of South Fishway.



Rock — lava basalt 91,000 cubic yards.

Report — Eimco's 105 Tractor Excavator was purchased for this job after several other types of equipment had been tried and proved either entirely unsatisfactory or incapable of maintaining the desired pace. The fishway (shown above) was excavated in lava basalt (typical of this section) with its characteristic flows and faults. Blasting in the faulted zone left many large blocky pieces of rock.

The method successfully used to excavate the shot rock from the 1500 foot long by 15 foot wide fishway channel which was 16′ deep at one end and 60′ deep at the other is as follows: A 6 yard rock ship was lowered by a crane and held in place by a Koehring Dumptor. It was loaded by the Eimco and hoisted out. "This method proved to be the most satisfactory and economical. Both machines took a terrific beating to get this job done, but to this day no one has been able to suggest a better way of doing it."

When it comes to bidding on a tough job — here are a few factors to consider: (1) Cubic yards of tough rock loaded per hour given proper haulage equipment — the Eimco will handle on a comparable basis with boom type shovels 1½ to 2 times its bucket size. (2) Cost—compare the prices. (3) Maintenance — compare these costs on jobs similar to your own. (4) Versatility — The Eimco moves quickly, easily. (5) Labor — compare the productivity per man hour.

Yes! You too will find Eimco's best for the tough jobs.

THE EIMCO CORPORATION

Salt Lake City, Utah—U.S.A. • Export Offices: Eimco Bldg., 52 South St., New York City

New York, N. Y. Chicago, Ill. San Francisco, Calif. El Paso, Tex. Birmingham, Ala. Duluth, Minn. Kellogg, Ida. Baltimore, Md. Pittsburgh, Pa. Seattle, Wash. Pasadena, Calif. Houston, Texas Vancouver, B. C. London, England Gateshead, England Paris, France Milan, Italy Johannesburg, South Africa

B-178

48,500 Pounds of payload yet this two-trailer hopper train weighs only 29,500 pounds empty. The big capacity pays off for Motor Freight, Inc. of New Philadelphia, Ohio. This operator uses the equipment to haul coal under contract from mine to a plant 54 miles away.

Deadweight is cut by utilizing USS COR-TEN Steel, a high strength low alloy steel containing nickel, produced by United States Steel Corporation, Pittsburgh, Pa. COR-TEN provides 50% higher effective strength than structural carbon steel. Steels of this type give wide play to the skill of engineers.

62% of gross weight is payload!

Designed to utilize high strength low alloy steel containing nickel

"OUR FUTURE TRUCKS will be of the same material..." declares Robert Ress of Motor Freight, Inc.

Lightweight structures with stamina to withstand impact, battering and abrasion are readily fabricated from high strength low alloy steels containing nickel.

Also, when attacked by atmospheric corrosion, most steels of this type offer 4 to 6 times the resistance of structural carbon steel. As a result, the nickel alloyed steels retain a high degree of original strength over years of use.

Which means much more life per dollar invested.

What's more, use of high strength low alloy nickel steels increases revenue per ton mile. In either of two ways.

1. You can cut deadweight and increase payload without decreasing structural strength.
2. You can increase payload capacity without increasing total weight.

Take advantage of these modern metals. Design your equipment to earn more profit. Moderate in cost, high strength low alloy steels containing nickel along with other alloying elements are produced under a variety of trade names by leading steel companies. Learn more about these steels . . . send for a copy of "Nickel-Copper High Strength Low Alloy Steels." Do it now.

THE INTERNATIONAL NICKEL COMPANY, INC. 67 Wall Street New York 5, N.Y.

New Firm to Conduct Uranium Ore Buying

Lucius Pitkin Inc., a firm of metallurgical chemists and consultants, took over from American Smelting & Refining Co. on February 1 as contractor for the uranium ore buying and concentrate receiving function of the AEC in the western U. S. The Pitkin firm, which has been under contract to the Commission's Division of Raw Materials for a number of years, will operate the AEC's ore buying stations in addition to the concentrate sampling plant and assay laboratory at Grand Junction, Colo.

Output of Titanium Ore to Increase

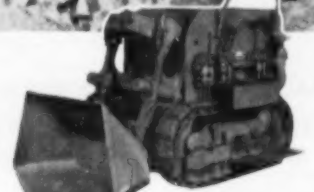
Metal & Thermit Corp. has purchased an 800-acre tract near Montpelier in Hanover County, Va., and will shortly begin construction of a plant for processing ilmenite and rutile ores from this property. The plant, expected to be in operation late this year, will include crushing, gravity ore dressing, and magnetic separation units. . . . At Tahawus, N. Y., National Lead Co. is increasing the capacity of its ilmenite mine and mill by 25 pct this year. During 1955 the firm reported the discovery of a new orebody at this property which may boost reserves by 100 million tons.

Oil Shale Mine to Become Underground Laboratory

Aided by a $1 million appropriation, the U. S. Bureau of Mines oil shale property at Rifle, Colo., will become an experimental mine where research on rock movements, ventilation, drilling, blasting, and automation will be conducted. Safety will be highlighted with special devices planned to warn of impending roof falls. The project is designed to lead to the development of safer and cheaper methods of mining oil shale.

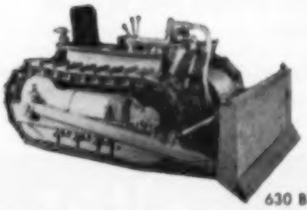
Iron Ore Project for Michigan

A new company, organized by the Cleveland-Cliffs Iron Co., will be the first major unit for concentrating and pelletizing the low grade jasper iron ores of the Marquette Range of Michigan's Upper Peninsula. Also participating are Inland Steel Co., Jones & Laughlin Steel Corp., Wheeling Steel Corp., and International Harvester Co. The new firm, Marquette Iron Mining Co., has leases on the Republic and the Empire mines, where beneficiation plants are being built. At Republic, recovery will be by flotation, while at Empire ores will permit a magnetic separation. An agglomeration plant, under construction at Negaunee, will pelletize the 60 pct concentrates. The plant, at the site of the discovery of iron ore in the Lake Superior district, is being built in ½ million ton units, the first of which will go into operation later this year. It is expected to have an eventual capacity of 2 million tons of pellets.



630 Loader

630 Tractor



630 Bulldozer

NEW SHAFT SINKING TECHNIQUES

Shaft sinking contractors have developed a new technique for loading out the broken rock in the bottom of the shaft by using the Eimco 630 crawler.

The method consists of lowering the Eimco immediately after the blast and proceeding to muck out the round into 3-yard tubs. Time per loading is approximately 2½ minutes but will probably improve as the crew gets more familiar with the new equipment.

The difference in the techniques is in the method of shooting the bottom which leaves the bottom in the shape of a "V" which provides a "face in the shaft for the loader operating on an incline.

The introduction of the Eimco 630 in shaft mucking eliminates the necessity of using expensive single purpose equipment, special framing in the shaft, temporary guides and other special setups. When the shaft is completed the Eimco is ready to work in the drifts on production loading.

Send for more information on the Eimco 630.

THE EIMCO CORPORATION

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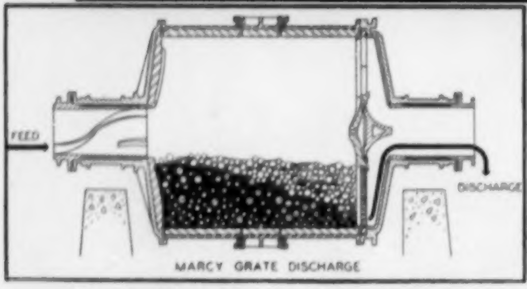
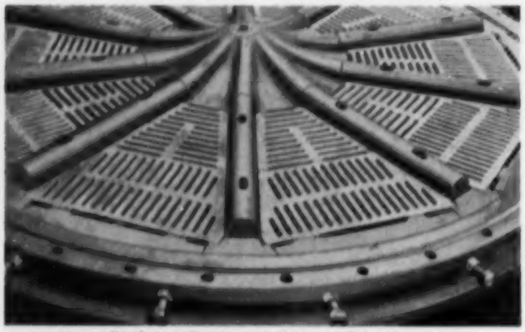
B-181

how MARCY EXPERIENCE

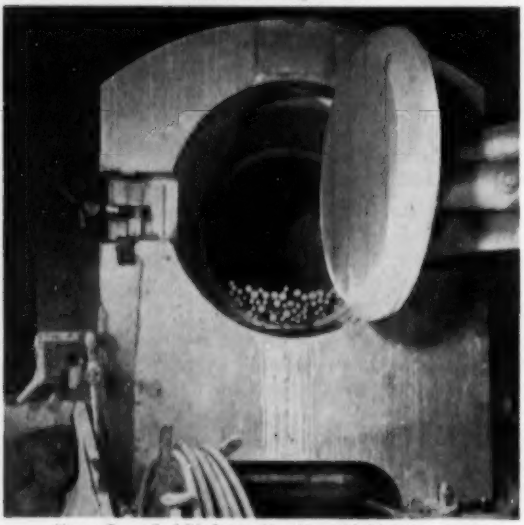
increases grinding mill output up to 50%



Marcy Low-Pulp-Line Grinding, with Grate Discharge.



Marcy Discharge Grates with End and Side Clamp Bars and Center Discharge Liner.



Marcy Open End Discharge Housing, with Plug Door open.

After years of research in the field of grinding Mine and Smelter made the first ball mill, in 1915, incorporating the Marcy principle of grinding . . . *"rapid change of the mill content is necessary for high efficiency"*

The "rapid change of mill content" is accomplished by maintaining a low pulp line through use of the Marcy full-grate discharge in ball mills and the Marcy open end feature in rod mills.

Early operating *experience* proved the effectiveness of this basic principle in giving greater output at lower KWH per ton than other type mills. Refinements in design and construction, which can come only from *experience*, have continually improved both the mechanical and metallurgical performance of Marcy Mills.

This *experience* by M&S has resulted in production of Marcy Mills which, by actual operating data, have proved their ability to have up to 50% greater capacity than other type mills.

PROVED ADVANTAGES OF MARCY LOW-PULP-LINE GRINDING...

- **eliminates wasteful cushioning action of high pulp level.**
- **provides an active, effective grinding mass to act on particle size reduction only.**
- **there is a faster migration of fines than oversize particles, thus less overgrinding.**

YOU CAN HAVE THE ADVANTAGE OF ALL THIS GRINDING *EXPERIENCE*... JUST WRITE, CALL OR WIRE...

The Mine & Smelter Supply Co.

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J&L Steel Options Large Iron Ore Deposit Near Labrador

Montreal February 20—Quebec Cobalt & Exploration Ltd. has signed an agreement with Jones & Laughlin Steel Corp. of Pittsburgh. The terms provide that for a period of two years and a guaranteed minimum expenditure of $250,000, Jones & Laughlin, the fourth largest steel producer in the U. S. will explore iron ore properties of Quebec Cobalt & Exploration Ltd. in the Mt. Wright area of New Quebec, 190 miles north of the port of Seven Islands and 40 miles west of the Hollinger-North Shore Railroad, which serves the Knob Hill-Labrador iron deposits approximately 180 miles north of Mt. Wright.

Jones & Laughlin will study plans for construction of a beneficiating plant if development work meets present expectations within the two-year period. Preliminary work conducted by Quebec Cobalt since 1952 has outlined approximately *one billion* tons of 32 pct iron ore. Most of this is a *coarse hematite* with lesser amounts of magnetite. Jones & Laughlin work during the 1956 and 1957 seasons will consist of drilling and testing this large deposit.

The Operating Agreement provides for a lease whereby minimum royalties of $50,000 a year for a period of two years will be paid to Quebec Cobalt. Thereafter minimum payments will be increased to $100,000 yearly.

When the property is in production, a royalty of 60¢ per ton of concentrate will be paid subject to an upward revision of 5¢ per ton for each $1 per ton advance in the Lake Erie price of iron ore, the base price being $10.10 per ton. In addition, there is provision for a royalty of $1 per ton on direct shipping ore that may be mined with an escalator clause for increased royalty in the event iron prices advance. Royalties are subject to revision based on operating profit. Any minerals other than iron which may be found on the property and are not a part of the iron deposits are reserved for Quebec Cobalt & Exploration.

Map shows site of Quebec Cobalt & Exploration Ltd. property optioned by Jones & Laughlin Steel Corp., as well as relative position of deposit to J&L's Pittsburgh steelmaking operations.

Preparations are now going forward to transport drills and other equipment to the mine. In the event Jones & Laughlin's work meets with expectations and plans to equip the property for production are made, New Quebec will see the beginning of a new era with an iron ore beneficiating industry.

In its release, Jones & Laughlin pointed out that if the property is proven it will represent a major addition to the company's iron ore reserves. "Announcement of the optioning of the large deposit and the impending start of drilling is an indication of the continuing search for iron ore reserves by Jones & Laughlin."

"The reserve is estimated by Quebec Cobalt at approximately 250 million tons of concentrates. If the deposit is proved by the impending J&L exploration, the ore will have to be beneficiated to produce high grade ore containing 60 to 65 pct iron, probably in the form of pellets. Such beneficiation will require a large investment in plant and equipment to process the ore."

Anaconda Looks to Expanded Chilean Operations

Plans for the "greatest and most important development of copper mining in Chile since 1914" were unveiled recently by Roy H. Glover, board chairman of The Anaconda Co. To Chilean President Carlos Ibanez del Campo, Anaconda's subsidiary, Andes Copper Mining Co., outlined a proposal for mining and beneficiating the copper ores of a district some 18 miles north of Potrerillos. Proposed under the provisions of Chile's new copper law, the plan, if approved, will result in the investment of a total of $52.95 million, according to preliminary estimates.

Exploration work has been conducted since 1952 on the ore bodies located near the Indio Muerto Mountain. Intensified drilling has been confined to an area called Turquoise Gulch. A limited amount of drilling in only a small part of the mineralized area, has already developed a reasonably assured ore tonnage of approximately 78 million tons with a grade of approximately 1.6 pct copper. A large amount of underground development and additional drilling will continue to prove additional ore believed to be in the district.

Projected installations and planned revisions of the nearby existing plant of Potrerillos are expected to provide an ore capacity of 25,000 tpd. The estimated annual capacity for production is about 100,000 short tons of fine copper.

The Andes Copper presentation revealed that the ores of the present Potrerillos mine would be exhausted in four to five years, and that a like period would be required to bring into production the ores of the Indio Muerto Mountain district. An additional period of one to two years will probably be needed to bring the new mines up to full production. This ore from the Indio Muerto Mountain district can be delivered as a fine crushed product to the existing Potrerillos plant which, after necessary revisions and re-equipping, can handle this ore.

Brief Description of New Project

The present plans provide for the development of the deposit in the Turquoise Gulch area as an underground mine. Into this mine there will be driven a main haulage tunnel which will be electrified for the haulage of the ore to the head of the surface railway. There will be an inclined shaft driven into the working levels of the orebody from a point near the Turquoise Gulch area to be used for service, personnel and materials. Ore bins and coarse crushing plant will be constructed underground.

The surface installations proposed will consist of transmission lines and water lines into the property, the construction of approximately 16 miles of meter gage railway connecting the tunnel portal of the Turquoise Gulch mine to the fine crushing plant site in the Pastos Cerrados Canyon below Potrerillos at an elevation of approximately 2400 yd; a belt conveyor system about 3 miles long to deliver the finely crushed ore to ore bins at the main reduction works of the Potrerillos plant some 770 yd above the fine crushing plant; revision of the reduction plant installations at Potrerillos.

Further Plans Include Molybdenum

In addition to the $53 million Indio Muerto Mountain project presented today, other proposals for expansion in Chile will be made in the near future. Chile Exploration Co. will shortly make a presentation un-

(Continued on page 266)

MILLIONS of healthy, valuable young trees are growing in the nation's largest producing area of stripped coal. Mining companies in Pennsylvania and Ohio through reclamation efforts have improved game habitat, increased underground water supply, and prevented erosion on thousands of acres.

According to G. Albert Stewart, executive secretary, Pennsylvania Open Pit Assn., Philipsburg, Pa., "A lot of stripped surface plantings are today more valuable than adjoining acreage growing valueless species." Pine, locust, and other soft and hard woods now grow where wild cherry, june berry, devil's club, and other short-lived comparatively worthless trees grew before stripping.

Credit in Ohio for the success of many of these plantings must go to the Ohio Reclamation Assn. Its membership consists of almost 50 pct of the strippers in Ohio, mining 75 pct of the tonnage of the state. Records of the Ohio Div. of Forestry show purchases from state nurseries by coal companies since 1910. Tree plantings on strip land have shown a survival rate of approximately 85 pct. Damming of the last stripping cut made by mining operations has helped raise the water table in Ohio and has increased the state's water supply. More than 14 species of fish have been found in strip mine lakes, and muskrat, raccoon, and beaver have been seen in the lake marshes.

In Pennsylvania the Harmon Creek Coal Corp. at Burgettstown has reclaimed stripped land with astounding success. Although the firm started experimenting in the 1930's with reclamation of the land from which the company had taken coal, full-scale backfilling was not started until 1944. The first year 50 acres were graded and 14,500 trees planted. Initial plantings were made on the worst possible land with the idea that if these lasted plantings would grow on any of the areas. The company has now planted more than 586,000 trees.

Growth in the heavily strip-mined central Pennsylvania area is difficult because of the heavy concentration of sandrock and shale in this area. However, in the Allegheny areas reclaimed land planted in trees is considerably greater than in the eastern Ohio and southwestern Pennsylvania areas. Approximately 65 pct of the land reclaimed is planted in valuable pine and locust and other soft and hard woods.

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BEGINNER'S luck is holding true with uranium. An AEC spokesman recently told the Grand Junction Geological Society that inexperienced uranium prospectors have discovered more uranium than professionals have found. Amateurs look where experts know it can't be found. However, the AEC is suggesting to would-be uranium miners that they might have better luck somewhere other than the already heavily staked Colorado Plateau. The AEC is pointing to increasing reports of discoveries in Oregon, Washington, Texas, and California. Since then *Uranium Information*, the Denver weekly, has reported several off-Plateau discoveries that show promise. One of these is within the boundaries of the Garden of the Gods, just west of Colorado Springs. Ore is already being shipped from another eastern slope Colorado mine in the mountains northwest of Loveland. In Nevada pitchblende has been found on a group of claims 3 miles south of Austin in Lander County. Earlier exploration of the property showed the presence of secondary uranium ores. The pitchblende is 200 ft under the ground.

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DOUGLAS McKAY, Secretary of the Interior, recently presented his annual report on the U. S. Bureau of Mines to the President. Listed were more than 50 attainments during the 1955 fiscal year. Taking a visual hop, skip, and jump through it, here are a few of the many achievements:

A USBM-developed planer, a mechanical mining machine that can be operated by remote control, was tested successfully in an area of a Montana phosphate mine that had been abandoned as unsafe for conventional mining methods.

Studies went forward on the preparation and properties of artifical abrasives such as carbides, silicides, borides, and nitrides, to determine their suitability as substitutes for imported industrial diamonds.

In cooperation with the AEC, the USBM began research to determine whether nuclear energy can be utilized successfully in gasifying coal.

A safety device called a roof-bolt compression pad was developed. This promises to reduce even further the risk of failure of bolted mine roof. Although 26 pct of the U. S. coal production comes from roof-bolted areas, only two of the 177 roof-fall fatalities in bituminous coal mines in 1954 were due to failure of bolted roof.

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WANT to drill an oil well? If you're trying to save drilling costs, stay away from the Louisiana offshore area. In 1953 drilling a well there averaged $300,000, six times more than the average well on dry land. This figure is from the *Joint Association Survey*, made by the American Petroleum Institute, the Independent Petroleum Assn. of America, and the Mid-Continent Oil & Gas Assn. The study is based on 1953 figures, the most recent ones available when the survey started. That year U. S. oil men and oil companies invested an average of $50,000 in each of the 49,279 wells drilled in this country. The study also reveals how much the cost increases with the depth drilled. Drilling costs per incremental foot in the 3750 to 5000-ft range averaged almost $13, yet they ran close to $106 per ft beyond the 15,000-ft level.

Meanwhile McPhar Geophysics in Canada is claiming that an excessive drilling program can be eliminated. This company makes a transmitting and receiving set that can be lowered 3000 ft into a 2-in. diam drillhole to search out 300 ft in all directions for orebodies. The set consists of a series of tubes inserted in a probe.

SIXTEEN companies prominent in the mining field have been certified as excellently managed by the American Institute of Management. Of these companies, 13 are receiving the award for the sixth consecutive year and three for the fifth year. The AIM uses ten factors in auditing a management: economic function, corporate structure, health of earnings, service to stockholders, directorate analysis, research and development, fiscal policies, production efficiency, sales vigor, and executive valuation.

In the six-year group are: Freeport Sulphur Co., American Metal Co., Consolidated Mining & Smelting Co. of Canada, Homestake Mining Co., Hudson Bay Mining & Smelting Co. Ltd., International Nickel Co. of Canada, Kennecott Copper Corp., National Lead Co., Newmont Mining Corp., Noranda Mines Ltd., Phelps Dodge Co., Texas Gulf Sulphur Co., and U. S. Smelting Refining & Mining Co. Those honored for the fifth year are: M. A. Hanna Co., The New Jersey Zinc Co., and St. Joseph Lead Co. This places the 16 among the 408 U. S. and Canadian firms cited by the Institute for 1955. Copies of the year *Manual of Excellent Managements* are available to nonmembers for $20.00 from the AIM, 125 E. 38th St., New York 16, N. Y.

ESSO now has a map with north on the bottom. It's to be used for a trip to Florida. But all indications are that the U. S. Geological Survey, now celebrating its 77th anniversary, is keeping north on the top of a map. In 1955 the USGS published some 2669 new and reprinted topographic maps. There are now topographic maps of good quality for about 37 pct of the total area of the 48 states. Total area of new mapping during the report year is about 4 pct of the continental U. S. At this rate, it will take another 20 years to map the entire U. S.

Also in the annual report the USGS cited the Idaho Minidoka Project, known as the North Side Pumping Div., as one of its most significant ground water studies. It is the first major Federal reclamation development based on ground water.

Drilling on public lands during 1955 included the spudding of 1413 wells and the completion of 1352 wells, of which 937 were productive of oil or gas. In all, 21,758 wells, including 12,433 capable of oil or gas production, were under supervision on June 30, 1955. Production was appreciably greater than in 1954 with royalty returns to the U. S. of about $39,222,638.

The USGS supervised 1813 mining properties and 110,577 oil and gas properties in Federal, Indian, and *acquired* lands. Total value of production amounted to approximately $523,753,229.

ANYONE looking for a needle in a haystack might run over to American Steel & Wire Div., U. S. Steel Corp., in New Haven, Conn., or Trenton, N. J., and see about borrowing a new electronic detector. This device can detect a defective single wire indiscernable to the naked eye in a group of more than 222 individual wires stranded together. Basic principle of the detector is the creation of a magnetic field by constant power source in the core of a sensing coil. Irregularities in the wire rope change the density of this magnetic field or flux. These defects are measured and calibrated as output signals that are in excess of a predetermined range. Pick-up coil then energizes electronic relays. These flash a warning light, sound an alarm, automatically spray paint on the spot, stop the machine, or presumably shout "Hey, Charlie!" to the operator who eliminates the faulty section from the production line. So far some 1.5 million ft of rope products have been inspected by this device. It has not displaced any workers.

A *DECADE of Progress in Canadian Mining* (1945-1954), Part II, appeared in the *Bulletin* of the Institution of Mining and Metallurgy for January 1956. The author, H. A. Graves, is a mining engineer with the Mineral Resources Div., Mines Branch, Ottawa. He is writing about his country, but he could be writing about ours. Mining in the U. S. is vastly indebted to technological advances in Canadian mining.

He finds that "the achievements of the atomic age, though justifying the expectation of many new developments, tend to increase the dependence on minerals rather than to relieve it.

"To an increasing extent production of metals and minerals must come from lower-grade ores requiring the expenditure of more man-hours, power and materials per unit recovered, as well as larger capital investment. Thorough exploration of potential mineral resources must be continued in the search for new orebodies . . .

"Although the average grade of ore being mined has been declining for 50 years, and wage rates and price of supplies have been greatly increased, the mining industry has continued to grow and to prosper. The rise in metal prices has been a factor in this prosperity but important too is the fact that technical developments have advanced to a point where minerals can be mined under the most severe conditions . . ."

Anaconda, Cont'd.

der the provisions of Chile's new copper law, to build and equip a plant to recover molybdenum from copper ores mined at Chuquicamata. He predicted substantial additional gross revenue within the next several years because the molybdenum content of the copper ores will increase as the depth of the Chuquicamata pit is increased. He also disclosed that ores so far proven at the Indio Muerto Mountain district also contain molybdenum, and in quantities higher than found in the Chuquicamata ores. As further work proceeds on the Indio Muerto district, a proposal to invest in plant and equipment to recover molybdenum from these ores will also be made. Mr. Glover pointed out that recovery of molybdenum both at Chuquicamata and at the new project, will be strictly a by-product of copper production.

He announced that the Chile Exploration Co. will shortly make presentation of a project to partly revise its electrolytic tank house from an extractive operation to an electrolytic refining operation. Also that the Andes Copper Mining Co. contemplates making a similar proposal at a later date. The changes would place in effect processes whereby copper now produced in the form of blister copper would be refined in Chile into the electrolytic form.

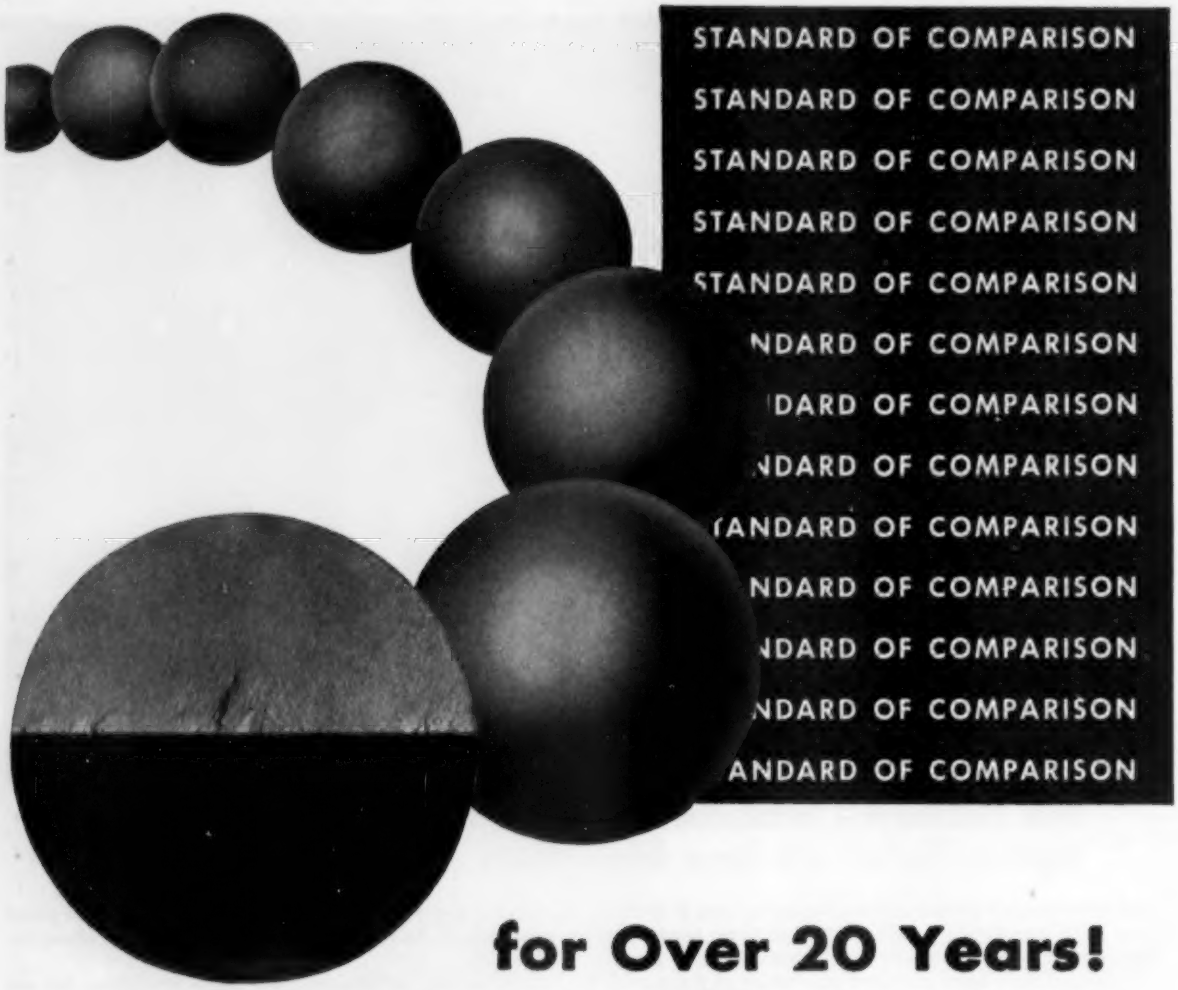
Mr. Glover reiterated his "unlimited confidence in the future of Chile." He called attention to the fact that Anaconda subsidiaries will have proposed, since the enactment of the new Chile copper law, investment programs totaling $100 million or more. This, combined with the $126 million recently invested in the sulphide plant at Chuquicamata, totaling a quarter of a billion dollars, will easily make the largest investment program ever undertaken in Chile.

Leighton Heads Geological Institute For 1956

New officers of the American Geological Institute for 1956 are as follows: M. M. Leighton, president; J. L. Gillson, Vice President; G. L. Jepsen, Secretary-Treasurer. The executive committee also includes ex officio the past president, E. A. Eckhardt, and the chairman of the Div. of Earth Sciences of the National Research Council, R. J. Russell. The executive director of the AGI since October 1 has been Robert C. Stephenson, for the last year Chairman of the AIME Mining Branch Council.

ESPS-San Francisco Completes 30 Years

The AIME joins with other cooperating engineer organizations that sponsor the Engineering Societies Personnel Service in extending to the San Francisco office of ESPS congratulations and best wishes. This office has completed 30 years of service to engineers of the western states. Of the 8000 western engineering positions filled with the assistance of San Francisco ESPS, 1500 have been for mining, metallurgical, or petroleum engineers in extraction, reduction, processing, and other fields. AIME Members of the San Francisco Section who have served on the local advisory committee include W. H. Grant, R. A. Kinzie, S. Bordon, F. Collins, and the present chairman, E. W. Bullard, president, E. D. Bullard Co. Edward H. Robie, AIME Secretary Emeritus, and E. J. Kennedy, Jr., Assistant Secretary, AIME, serve on the national board of directors for ESPS.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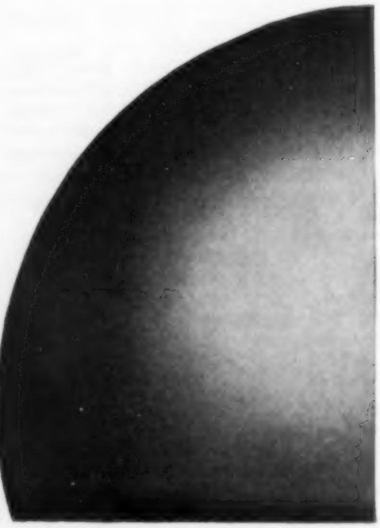

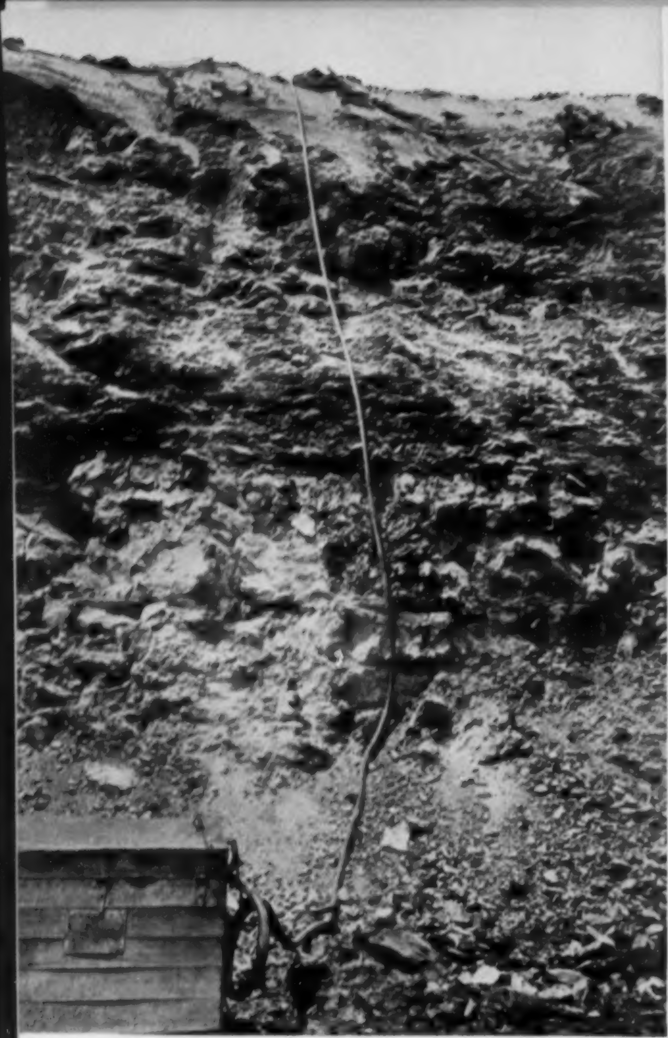

SPECIALLY DESIGNED for rugged conditions like this, Anaconda butyl-insulated Mine Power Cable delivers maximum service and safety.

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We proved this power cable in our mines

Being miners ourselves, we know mining problems. As both miner and cable manufacturer we're able to do something about them—by *designing* cable to meet problems only miners can know . . . by testing this cable in *our own mines* under actual field conditions.

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Our firsthand mine experience has helped us build a sturdy cable that cuts down-time. Butyl insulation gives this cable long-aging characteristics, improved resistance to moisture, ozone and heat. Neoprene jacket—rugged, tough—has real flexibility, and resists rock-cutting, impact, flame, sun, and corrosive mine water.

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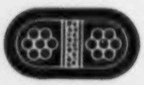
FLAT-TWIN CABLE



Improved stranding, new insulation, new grounding wire, and neoprene jacket make this a superior cable for shuttle cars, continuous miners, loaders, drill trucks, cutters.

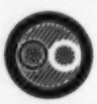
POWER CABLES

Anaconda Types W & G are rugged, sturdy and long-lived. Used for mine power, shovels, continuous miners, loaders, drill trucks, cutters.

SHOVEL AND DRILL CABLES

Securityflex* Types W and G are used with small shovels, self-propelled drill trucks, pumps and a-c mining equipment. For higher voltages, Type SH cables (shielded) are recommended.

SECURITYFLEX CORDS



Unexcelled for strength, wear resistance and long life. Type SO cords (heavy-duty) provide superior service on remote control and hand drills.

- TROLLEY WIRE
- FEEDER CABLE BARE OR INSULATED
- TELEPHONE WIRE
- SHOT-FIRE CORD
- WELDING CABLE

MINING engineering

DRIFT

THE Presidency of this great organization is the kind of task that warms the heart. For it, I am deeply appreciative. I accept the challenge. I accept the responsibility. But I do so with a realization that measuring up to the standards of those who have preceded me will be a task of large proportions.

There is no better reminder of that sobering fact, I think, than the anniversary volume which AIME published back in 1947. It recorded the great moments of three-quarters of a century of progress in the mineral industry. It also paid a warm tribute to the men who founded AIME, who guided it through its formative years, and who finally handed down to us the mineral engineers of today a professional organization of great strength and vigor.

Before coming to this meeting, I again thumbed through that anniversary volume and I could not help noticing the size of the shoes that we of today and tomorrow must fill. In olden times, according to the Bible, there were giants in the earth. Look over our anniversary book, that anthology of achievement, and you will see that there have also been giants—giants of engineering—in this institute.

Our founding group, for example—those 22 engineers who met at Wilkes-Barre, Pa. in May of 1871 and organized AIME—stood head and shoulders above the common run of men. It is true, of course, that their fields of mining and metallurgical engineering were still in their infancy. Only a few decades before, there had been only two recognized divisions of engineering—civil and military. Up until the 1850's, there were just two schools in America where one could get an engineering degree—at West Point and at Rensselaer Polytechnic Institute. Not a single state had yet enacted an engineering practice law.

But these 22 men had the pioneering instinct. They not only had faith in engineering as the profession of progress. Their faith found solid expression in their building of AIME and in the two main objectives which they set up as organizational goals:

"First, the more economical production of the useful minerals and metals. Second, the greater safety and welfare of those employed in those industries."

It is no small credit to the vision of AIME's founders that these objectives, as currently set forth in our constitution, remain basically unchanged.

To this day, their great faith, their great vision, continue to stand out as dominant characteristics of AIME. From the time of David Thomas, that great ironmaster who was our first president, down to and including the past year under H. DeWitt Smith, the Institute has built, added to, and conserved a tremendous heritage of engineering knowledge. It has built, added to, and conserved an equally important legacy of professional concepts. By so doing, AIME has not only advanced the art and science of engineering. It has also enabled each of us to better serve the public interest.

The Institute could never have accomplished these things under a leadership with a policy of standing still. Fortunately, however, its officers and directors, its committeemen and rank-and-file members, have always held to a guiding principle of continuous progress—the kind of continuous progress which started with prehistoric man's first use of fire and which has inexorably led, in our own time, to today's wide, wide world of atomic power. Their guiding principle, in short, has always been one of seeking new and better ways, under a profit-and-loss system, to convert the riches of mankind to the use of mankind.

In following this principle, the Institute has endeavored to see to it that the professional lives of each of us, as individuals, benefit as fully as possible from our group relationships in the society. Its actions have been positive ones. We have steadily increased our membership until today there are more than 26,000 engineers and engineering students on our rolls. Since 1911, we have chartered and sponsored the local sections which are now such a vital part of the organization. As our needs have grown, we have expanded our technical committees, our professional divisions, and our publication services.

Most important of all, perhaps, the Institute has provided a forum of inestimable value to all those men of engineering whose work has to do with our mineral resources. And this forum has been one with an open-door policy. In 1871, for example, when the Institute was founded, the petroleum industry was a relatively insignificant business mainly engaged in providing kerosene for the lamps of America. After Spindletop, however, when petroleum finally burst into real national prominence, AIME lost little time in opening its doors to the engineers of this growing new industry.

In 1913, the Institute set up a petroleum and gas committee under Captain Anthony Lucas, the discoverer of Spindletop. Nine years later, in 1922, it formed the Petroleum Division. Equal footing in the Institute has brought equal responsibilities and equal benefits. During the past third of a century, the presentation of papers on petroleum technology before AIME groups has marked some of the industry's most important milestones. Here again, I think, is another indication of the foresight and the pioneering instinct of this organization. AIME was founded by mining and metallurgical engineers, and they were under no obligation to take in their petroleum industry colleagues. They saw quite clearly, though, that we all are, in the final analysis, a definite breed of men. We are all working with mineral resources and converting them to the use and convenience of mankind. We share common goals, common problems, and common interests.

Let me say for the petroleum industry and I feel sure that past presidents Everette De Golyer, John Suman, and Mike Haider will agree with me that this mutual association has been tremendously valuable. It has been so valuable that had there been no AIME for us to join, it would have been absolutely necessary to have invented one.

This Institute, then, has come far in the last 85 years. It has made giant strides. But to say that the road has never been bumpy—or to say that it will never again be rough—would be to distort the facts. This being George Washington's Birthday,* we could very well draw an analogy, I believe, between the kind of problems that confronted him and our other forefathers when they founded our National Government and the kind of problems that have confronted those of us in AIME. They had different peoples and religions to merge into one political unit. We have had different fields of engineering to consider—fields of engineering that are close together in many basic respects but which, in other ways, are poles apart. Our forefathers had geographical problems—problems of varying conditions in different areas. We have had similar problems. The matter of the relationships between the federal Government and the various states was another knotty problem. On a smaller scale, we have had to work out similar functioning relationships between our National Headquarters and our many local sections.

We shall continue to have these problems. The field of mineral engineering, once relatively limited, continues to widen day by day. As it does, the problems of our organization become more numerous, increasingly more difficult to administer our affairs in such a way as to see that all members are served fairly and effectively. However, as long as we keep intact our common bonds, as long as we keep alive that broad devotion to the engineering profession, no problem in AIME should prove too difficult for solution.

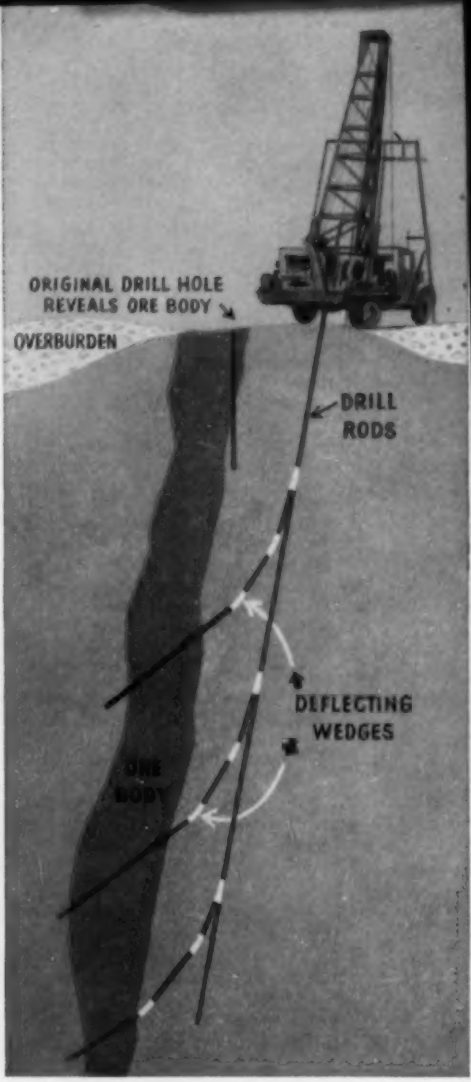
What can we, as individuals, do about these problems? By putting out a little extra effort—and by keeping the proper amount of tolerance, the proper appreciation for the other fellow's viewpoint—we can build a stronger AIME than ever before.

As your new President, I earnestly solicit your support in pursuit of that goal.

Carl E. Reistle, Jr.

* Originally presented at the AIME Annual Banquet Feb. 22, 1956.

Drill rods of Pittsburgh Steel are used with mobile drill rigs such as this Longyear jeep-mounted model in the world-wide quest for vital minerals.



Directional drilling tests skill of drill crew, strength of drill rods.

Hitting The Bullseye A Mile Underground

Pittsburgh Steel drill rods used by E. J. Longyear Company since 1890's to score impressive prospecting firsts.

Modern prospectors for vital raw materials are scientists. That's why geologists and engineers do the exploratory drilling for E. J. Longyear, the nation's number one prospecting organization.

Since the 1890's when E. J. Longyear himself drilled the first diamond bit into the rich iron ores of the Mesabi Range in Minnesota, Longyear prospectors have used Pittsburgh drill rods. They have accumulated an impressive list of firsts. Here are a few of them:

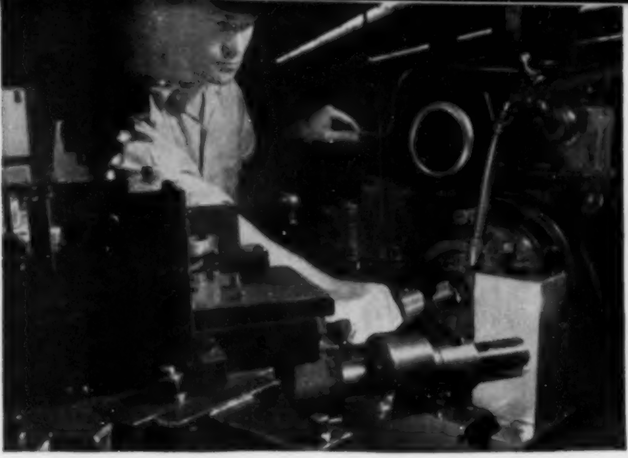
- **Nickel**—Discovery and development of the millions-of-ton reserves of the Falconbridge Mine near Sudbury, Canada, in 1916.
- **Copper**—First diamond drill exploration of the multi-million-ton reserves of the Roan Antelope and Mufulira Mines, Africa, in 1929.
- **Iron Ore**—First exploration of the half-billion-ton reserves in the Cerro Bolivar, Venezuela, in 1947.
- **Uranium**—First exploration of one of the world's largest (several million tons) uranium bodies at the Jackpile Mine, New Mexico, in 1955.

Here's how Longyear goes about finding strategic minerals:

After geological surveys, the earth's surface strata is pierced with steel drill rods equipped with diamond drill bits. Stabbing into the earth to depths of more than a mile, the drillers strike a target area that often is extremely small. As the drill bit advances it carves out cylindrical cores of the earth's structure. These cores are lifted out for study.

Drillers are under constant competitive pressure to reduce the cost per foot of hole drilled. Their success in keeping costs low determines the profit or loss on an operation, so drill rods are an important factor.

The usual "deep" drill hole will go between 3,000 and 5,000 feet



Finished drill rods (right) provide a tight joint for internal couplings after they are threaded (left).

although some run 6,000 feet and more. Drill rods are used in lengths varying from 2 to 20 feet depending upon the type and size of the drill rig and the depth of the hole.

At 6,000 feet the weight of "NW" size drill rods suspended from the drilling platform is about 17 tons. This is being rotated by a 50 to 75 horsepower engine at speeds up to 800 rpm.

For speed and low costs in drilling, then, the drill rod must come from steel with the physical strength to withstand tremendous torque loading, shock, abrasive wear and fatigue.

If the rod breaks under the abrasion, torque, or the fatigue wear it receives, an expensive "fishing" job is required to recover the drill rods and save the drill hole.

If the rod is not straight and concentric, it develops a "whip" and causes premature drill rod failure and excessive wear.

In directional drilling, the drill rods are subject to all normal physical requirements multiplied many times because the rods are forced around bends while being rotated.

In addition to these requirements, the steel for drill rods must have good machinability that will permit high speed precision machining at low cost.

Longyear uses Pittsburgh Steel's cold drawn seamless mechanical tubing in the .30-.40 carbon range, with an internal upset at both ends, normalized to remove residual stresses and assure complete dimensional uniformity. It provides a tensile strength of 65,000 psi and hardness of between 28 and 32 on the Rockwell C scale.

This tubing arrives at Longyear's plants in lengths that will make 5, 10 or 20-foot drill rods. The outside diameter for these rods ranges from $1\frac{5}{8}$ to $2\frac{3}{8}$ inches. They have a wall thickness of $\frac{5}{16}$ of an inch.

The internal upset enables them to be machined to provide a snug seat for the internal coupling that holds the rods together in a string, without weakening the wall. The upset on a $2\frac{3}{8}$-inch outside diameter tube is to a $1\frac{5}{8}$-inch inside diameter for a distance of 3 inches on each end. A slight defect in the quality of the steel can cause costly tool wear or broken tools.

Pittsburgh Steel meets all of these requirements consistently. Longyear can count on drill rods made from it for economy and reliable performance in the field. That's why you'll find it on almost every Longyear drilling operation.

If you have an application for seamless tubing, why not explore the advantages of reducing your costs and improving your products with Pittsburgh Steel tubes, carefully made to your specifications. A call to our closest district office today will bring prompt personal attention.

Pittsburgh Seamless Mechanical Tubing is also available from:

Baker Steel & Tube Company
Los Angeles, California

Chicago Tube & Iron Company
Chicago, Illinois

The Cleveland Tool & Supply C
Cleveland, Ohio

Drummond McCall & Co., Limited
Montreal, Quebec, Canada

Edgcomb Steel Company
Philadelphia, Pennsylvania

Gilmore Steel & Supply Co.
San Francisco, California

Earle M. Jorgensen Co.

Mapes & Sprowl Steel Co.
Union, New Jersey

Metal Goods Corporation
St. Louis, Missouri

Miller Steel Company, Inc.
Hillside, New Jersey

A. B. Murray Co., Inc.
Elizabeth, New Jersey

C. A. Russell, Inc.
Houston, Texas

Ryerson, Joseph T. & Son, Inc.
Chicago, Illinois

Solar Steel Corporation
Cleveland, Ohio

Steel Sales Corporation
Chicago, Illinois

Tubular Sales
Detroit, Michigan

Ward Steel Co.
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Ward Steel Service Company
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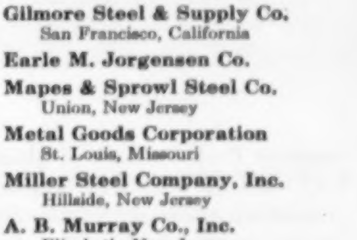

Seminole Rock Products of Miami, Fla., bought five CAT* DW15s with W15 Wagons on the basis of its previous experience with Caterpillar products. Dragline loaded with lime and coral rock, these units make a 1650-ft. haul (with short 10% adverse grade) over excellently maintained roads to the processing plant, and 2000-ft. return. After the Caterpillar DW15s had been on the job long enough to show what they could do, here is what General Manager L. G. Bunnell had to say:

"These DW15s have power to spare. With an average load of 15.2 yd. (45,600 lb. actual weight), haul and return time is only 4.09 minutes. The five units haul 358 cu. yd. per hour—estimated hauling cost is only 9.2¢ per yard. Thanks to their powerful Cat Engines, the DW15s started up from the dragline in second gear, high range.

"We are particularly satisfied with their extreme economy. In more than 1600 hours of combined operation, these five new Caterpillar DW15s have had no down time. Their rugged construction and trouble-free operation are important factors in cutting costs. Important, too, are their ease of maintenance and their ability to deliver full, foul-free power on No. 2 furnace oil.

"We like the high maneuverability of the Cat DW15 and W15 Wagon," Mr. Bunnell concludes. "This, with their dependability and power, speeds up our operation considerably."

Your Caterpillar Dealer—who provides skilled service and factory parts you can trust—will demonstrate the DW15 on *your* job. Give him a call soon.

Caterpillar Tractor Co., Peoria, Illinois, U.S.A.

CATERPILLAR*

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NAME THE DATE... YOUR DEALER WILL DEMONSTRATE

Stone Industry Production Problems Call for Research

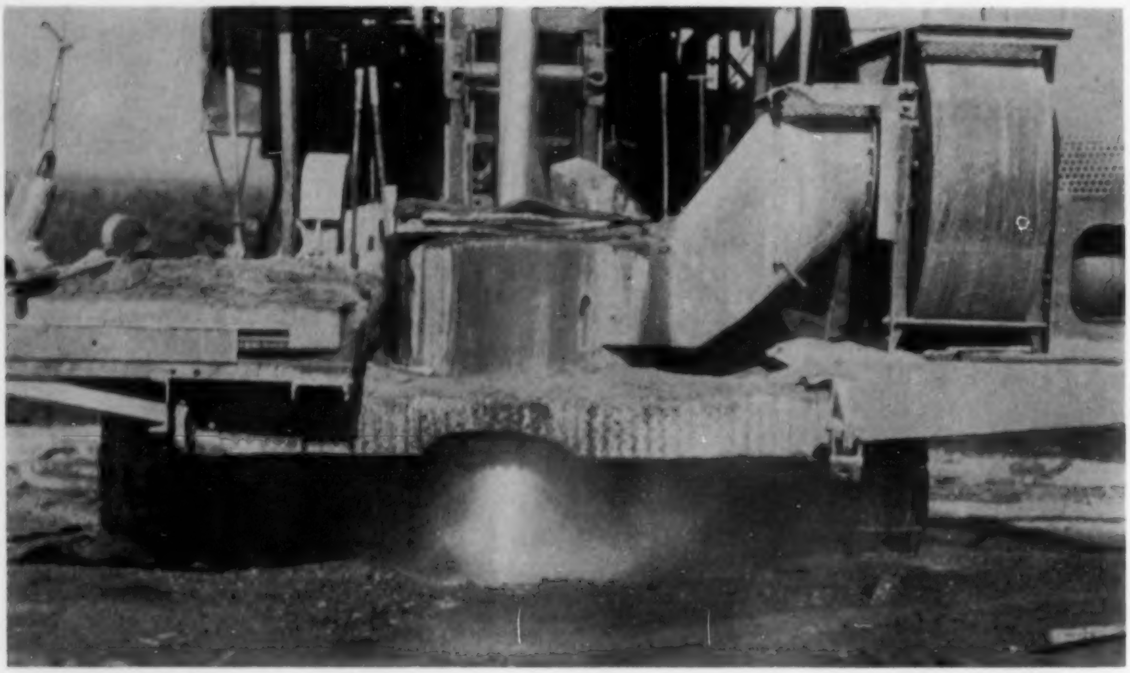
This operator suggests that through needed research the problems of the stone industry can be minimized by methods other than the trial and error approach.

by Nelson Severinghaus

CONSOLIDATED QUARRIES CORP. must conduct operations for an average sales price of $1.25 per ton, about the same price at which stone was sold 25 years ago when the dollar was worth twice what it is now. To maintain success in a competitive business, the company must solve the usual production problems, which will be discussed here in the order they occur in the flowsheet.

Most crushed stone quarries must first remove soil and decomposed rock. Although seasons of lower sales sometimes release regular quarry excavating and haulage equipment for this purpose, overburden is often too extensive for this method to be used. In such cases, special equipment must be assigned or the job may be contracted. As many earthmoving contractors have heavy duty specialized equipment well suited to overburden removal, they should be seriously considered before the job is undertaken. Pockets of clay beneath peaks of hard stone sometimes defy mechanized removal. Pick and shovel laborers at this point might be supplanted by more ingenious use of pumps and water or some other mechanical means. Where overburden is extremely heavy or difficult to remove, underground mining must be considered.

N. SEVERINGHAUS is Vice President and General Manager of Consolidated Quarries Corp., Decatur, Ga.

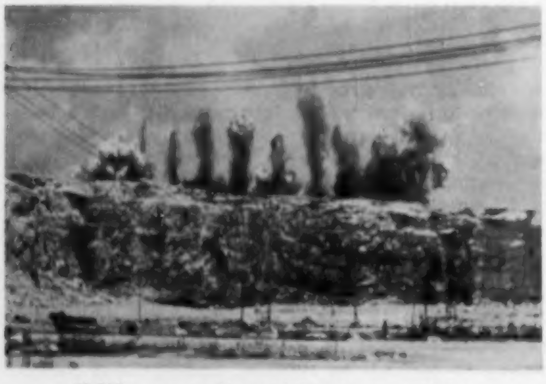


Start of a 7-in. jet piercing hole in granite. Consolidated found, however, that on a cost basis a 4-in. diam rotary drillhole gave best results.

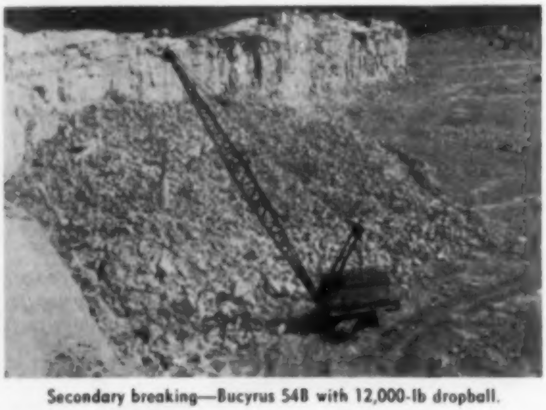
In drilling for placing of explosives, methods in use are almost as varied as rocks themselves. The goal here should be to obtain the best breakage possible consistent with reasonable drilling costs per ton of rock removed. The choice of size, spacing, and burden of holes for optimum results is indeed complex. This choice probably affects subsequent capacity and costs more than any other operating factor. Rock is broken down onto the quarry floor for a direct cost of 10¢ to 25¢ per ton. After that, four to six times that amount is spent before stone is finally loaded out for shipment. A 100 pct increase in cost of primary drilling and blasting might well be justified if plant capacity could be increased by one third with better breakage. It is easy to be penny wise and pound foolish when considering drilling.

Well broken stone in the quarry means greater capacity and less maintenance cost for equipment handling and crushing this stone. Working with good muck piles, shovels may well load out 50 pct more than when they are contending with a large proportion of oversize rocks that must be set aside for further breaking or even temporarily bypassed if they are too large to handle. Handling of oversize rocks at the primary crusher is generally the cause of most of the lost time for this equipment. During unproductive time due to these delays, other items of equipment, such as shovels and trucks, are idle though many cost items continue.

Extensive company experiments in drilling a rock body that breaks poorly have led to the use of smaller holes spaced closer together. In this operation it was found that on a cost basis a 4-in. diam hole gave best results with the equipment presently available. Fortunately newly developed drilling equipment and improved drill rods have made this possible without increasing drilling costs per ton. At present Consolidated Quarries is using a Gardner-Denver URMB-99 deep hole drilling wagon (4-in. piston percussion drill) and a Joy TM2 Challenger blasthole drill (rubber-tired, tractor-mounted 5¼-



50,000-ton quarry blast with 7-in. jet-pierced holes.



Secondary breaking—Bucyrus 54B with 12,000-lb dropball.

in. piston percussion type). The Gardner-Denver uses 1¼-in. hollow hexagon rods sleeve jointed in 8-ft lengths and 3-in. tungsten carbide bits. The Joy Challenger uses 2-in. hollow round rods sleeve jointed in 20-ft lengths and 4-in. tungsten carbide bits. Both rigs are drilling 90 ft, and drill life has been excellent.

This does not mean that most quarries will reach the same conclusion. Many rocks give entirely satisfactory breakage with 9 or 10-in. holes or even with tunnel blasts. Most of the granites will not do this. This problem of best size, spacing, and burden of holes must generally be solved by the trial and error method, but experiments can be guided by the experience of others. The emphasis here must be on overall results and not on cost of drilling alone.

In operating drills perhaps the most serious problem is the relative isolation of the operation and consequent difficulty of supervision. Most quarries are not large enough to warrant a drilling foreman. The logical alternative is to select the most able drillers available and then set up a system of daily reporting on performance. Where local labor relations allow, this may be coupled with an incentive pay system.

Most operators working with highly abrasive rocks have found that bit replacement and conditioning represent a major portion of drilling costs. Bit sharpness greatly affects rod failure, another large item of cost. With 4-in. tungsten carbide bits, Consolidated Quarries has found it advantageous to grind after each 20 ft of drilling. This obtains an average of ten regrinds or 200 ft per original new bit. With six pieces of 20-ft steel, more than 5000 ft of holes have been drilled without steel breakage. The question of proper footage between bit reconditionings needs study for each separate location.

Choice of blasting agents and methods of detonation presents another problem. There are any number of theories concerning best propagation speeds, method of detonation, delay patterns and intervals, bulk strengths, and optimum ratio of explosives to tonnage. Experienced representatives of explosives companies can be of great help here. They have worked with many types of rocks and used a wide variety of blasting methods. Conditions vary so greatly, however, that a certain amount of trial and error is necessary. Visual evaluation of blasting results is not always an accurate measure. If most of the oversize material is on top of the blasted pile, it can be reduced before shovels dig in and overall production results may be little affected. Prevalence of buried oversize, though not visible at first, will slow shovel and crushing plant production materially. Assuming an adequate supply of haulage equipment, shovel loading rates are probably the best measure of blasting results, and records should be kept of shovel performance.

It is entirely possible to obtain breakage that is *too good* from an economic viewpoint. Fines produced in blasting and crushing (if they can be sold at all) must generally be marketed at lower prices than coarser products. If too great a proportion is pulverized by the primary blasts, average sales price of entire production may be lowered so much that added production will not overcome the original handicap.

With quarry loading and haulage, the big problems are those of scheduling and maintenance. Quarry shovels are high-priced machines. They must be operated continuously at highest production rate possible to amortize them in reasonable time. To do this, quarry haulage must be entirely adequate. It is probably good economy here to have some excess haulage capacity. This excess serves as a storage bin between shovel and crusher to even out production of both. If one haulage unit is temporarily out for repairs, capacity of the whole plant is not lowered so drastically.

Quarry duty is rough on excavating and haulage equipment, particularly if machines are light. For those who operate 12 months in the year, maintenance must be scheduled during short off-duty periods. This requires a good supply of the parts which frequently fail or wear out. Repairs must be well organized and crews trained to do their work in the shortest possible time consistent with a good job. *Cleanliness or good housekeeping* is a necessity for proper maintenance. A steam cleaner is valuable for early detection of cracks and worn parts, particularly on shovels and trucks.

International TD14 bulldozer handling stockpile to tunnel conveyor. Bulldozers can be used profitably to extend stockpiles but are less efficient in raising the height.

Screening, storage, and loading facilities at Consolidated's quarry.

Breakdown of Time Lost in Routine

1. Lost motion in starting and stopping.
2. Refueling.
3. Replacement of parts such as dipper teeth.
4. Moving from one loading point to another.
5. Clearing roadways and sweeping in stray rocks.
6. Lubrication and operating adjustments.

At the primary crusher a number of problems are encountered that recur throughout the crushing and screening plant. One of these is dust control. Extensive water sprays at the dump point and at the crusher discharge are a great help, but too much water added at various transfer and crushing points will make fine screening difficult. Some operators are adding wetting agents to spray water. This aids in settling dust with much smaller amounts of water. In relatively compact plants it may be feasible to equip with a dust collection system. There are generally some markets for fine rock dust which will pay for part of the cost of collection.

If the primary crusher is a jaw, a feeder ahead of it is needed to obtain reasonably steady flow of material. Although a feeder will greatly reduce bridging in the mouth of the crusher, there are still delays from this cause. Adequate hoists with hooks or grabs to position or remove oversize rocks will cut delay time here. Care by shovel operators in laying back oversize rocks is a necessity for good crusher production.

Continuous flow on through the remainder of the plant with a minimum of labor requires transfer chutes of adequate size and slope. In handling abrasive rocks, Consolidated Quarries Corp. uses stone boxes in place of chutes wherever possible to reduce steel wear. When stone boxes are not feasible because of inadequate fall or the frequent need of cleaning out with change of size, liners of hard materials such as Nihard Cast Iron or abrasion resistant steel are used. Liners and parts with longer life mean lowered replacement labor costs.

Tramp iron can cause expensive delays and damage to belts and crushers, particularly secondary gyratories. It should be detected and removed as early as possible in the flowsheet. With heavily loaded belts, tramp iron can seldom be detected visually, and magnetic removal is not feasible, as nonmagnetic manganese steel dipper teeth are among the worst offenders. In recent years, satisfactory metal detectors for application to belt conveyors have been developed. With the use of relays, these can shut the belt down for hand removal of tramp iron. Thus a long-time problem seems to be nearing solution. Metallic fasteners must be eliminated from the belt before this device is used.

The problem of conveyor belt maintenance is largely one of preventing accidental damage and excessive wear due to improper loading. Good loading means a minimum free fall, direction and velocity close to that of the belt, and load well centered on the belt. Off center loads will cause belts to move sidewise with probable edge damage. These and other operating factors affecting belt life have been well covered in technical literature, and it is not necessary to go into great detail here.

Good performance from crushers depends largely on attention to the following details:

1) Exclusion of dust from bearings and gears by good maintenance of dust seals.

2) Use of the best bearing materials obtainable.

3) Proper lubrication by adequate circulating equipment and change of contaminated lubricant.

4) Maintenance of correct angles between crushing surfaces by replacing dished wearing parts or building them back to proper contour by welding.

5) Keeping bearing clearances low and all bolted joints tight.

Screening, particularly in the finer sizes, gives the most trouble in wet weather. In most crushed stone plants it is not feasible to use wet screening. Finer screens tend to blind badly with damp feed, resulting in lower feed rates and poorly sized products. Some operators have recently installed heated screens to overcome this difficulty, apparently with considerable success. Where particle shape is not too important a factor, elongated opening screen surfaces may be used to obtain a higher proportion of net opening to total area. Final recleaning with a good rinse just before loading out can take care of a considerable amount of poor screening farther back in the flowsheet. Material rinsed out at the Consolidated Quarries loading plant is sent to a sand washer and dewaterer, which produces some 35,000 tons of concrete sand a year. This requires settling out and impounding of −100 mesh material before wash water is released.

All too frequently, both space and facilities for stocking of finished stone are inadequate. Bulldozers are good extenders of stockpiles, but they do have limitations when raising the height of stone piles. Unless there is good recleaning after stockpiling, degradation of more fragile stones by heavy machines operating on piles may create a problem. Adequate stockpiles can do much to even out operations despite varying shipments, even to the extent of taking care of seasonal fluctuations. Stockpiles of finished materials must be financed, and the problem here is to balance cost of financing large storage against benefits derived in cost saving, steadier employment and service to customers.

One serious problem in the loading out area is to handle peak demands without excessive expenditure for spare men and spare machines. With quick changes in weather, loading requirements for a day

may be one fifth what they were on the previous day. At Consolidated Quarries loading by belt conveyors, which are fed from beneath stockpiles, has been the best answer. This system makes it possible to transfer unskilled men from other departments when loading is heavy.

Some mention has already been made of disposal of so-called waste products such as undersize from rinsing screens. More often dry fine screenings are in excess and must be stocked. It is important to remember that any excess products should be placed where they may be readily recovered and will not be contaminated with foreign matter. At one time the company had about 500,000 tons of $-1/8$-in. in a *waste* pile. Almost all this has been recovered and sold as new markets were developed. Handling, storage, and possible recovery of temporarily unmarketable products presents one more important problem in the stone industry. For years the Georgia Marble Co. accumulated reject blocks of marble along its railroad tracks. Now this company is sawing some of these blocks into slabs, splitting the slabs in a guillotine, and marketing a beautiful facing material for buildings.

Sometimes aggregates are needed in localities where the local rocks do not meet usual specifications without special treatment. An example of this would be a deposit which, when crushed, contained an excess of soft particles. If these soft particles can be removed, the remainder may well be a satisfactory aggregate. One possibility of separation lies in the fact that softer portions may be of lighter gravity than others. Heavy media separation has been successfully used in such cases. In other places attrition treatment has disintegrated and eliminated offending weathered minerals.

Another frequent cause of rejections of crushed stone is the presence of too many elongated or splintery particles. This is partly due to the characteristic fracture of some rocks, but steps can be taken to minimize the trouble. Certain types of crushers such as hammer mills produce a minimum of flats and splinters. If the offending rock is not so abrasive that crusher wear becomes prohibitive, this may offer a solution. In general, the lower the ratio of reduction in any one crusher and the nearer that crusher is to operating with *choke feed*, the more cubical the product will be. Various methods of separating cubical from splintery particles by screening have been tried, but none seems to have won wide acceptance.

A broader question facing all stone industry operators is the one of deciding on proper inventory of supplies and spare parts. The answer to this depends largely on local conditions such as relative isolation or availability of parts from nearby warehouse stocks. The advent of air freight with quick deliveries over longer distances has changed company decisions on necessary plant warehouse stocks. Urgency of operation of machines also affects the outlook on parts purchasing. If any department is over-equipped machines in that group can be out for repairs longer without critical damage to overall production. Proper storage and accounting for supplies demands close attention to avoid damage and loss.

These problems warrant research:

- Drilling and Blasting

  Development of a satisfactory method of evaluating results of investigation—execution of a program of carefully recorded experiments.

- Better Fragmentation

  Improvement through widest practical dispersion of explosives, weighing safety, timing, and costs.

- Lighter Equipment

  Development of lighter and smaller stone industry equipment that is easier to transport and erect.

- Tougher Materials

  Data on how hard material should be to resist wear from a given stone—materials that resist impact as well as abrasion.

- Product Sizing

  Better handling methods to lessen product degradation—stockpiling and loading methods that reduce size segregation—means to modify product size distribution without materially increasing plant costs.

Needed Research

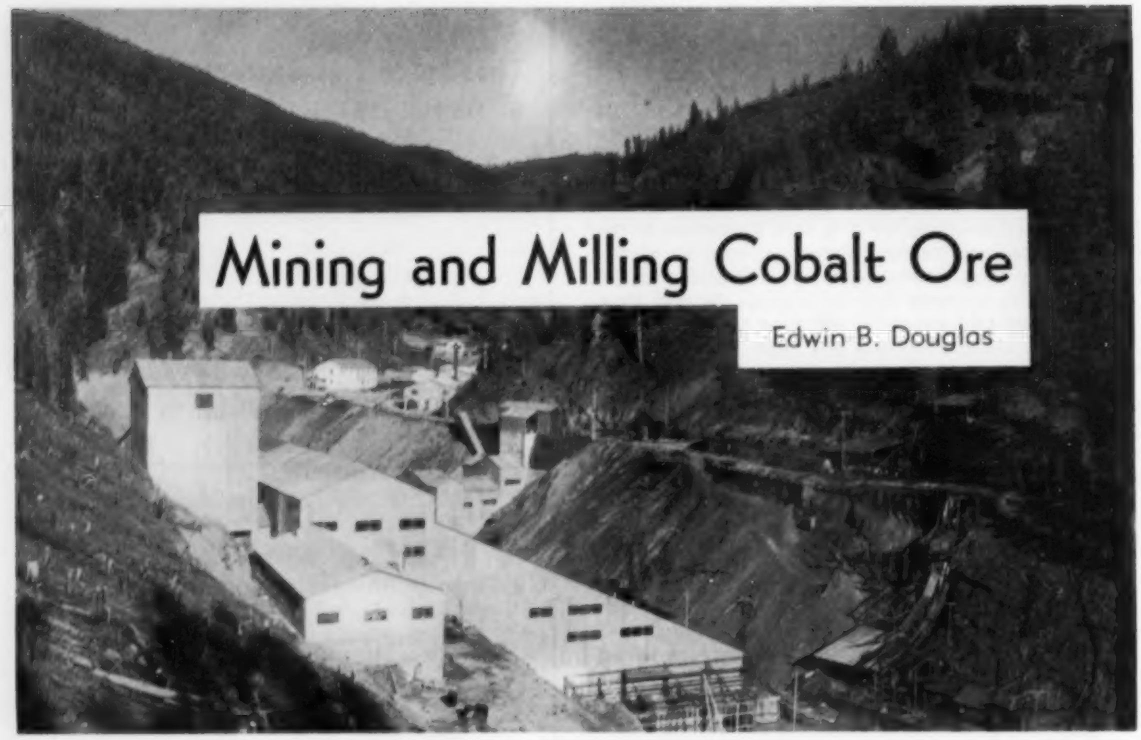
Industry and government now spend \$3.5 billion per year on research, ten times the amount spent prior to World War II. There is no reason to believe that the stone industry can progress without research when all other industries are acknowledging it to be a basic necessity.

This does not mean that each company must set up a research department and laboratory. Research is defined by Webster as "critical and exhaustive investigation or experimentation having for its aim the discovery of new facts and their correct interpretation, the revision of accepted conclusions, theories or laws in the light of newly discovered facts or the practical application of such new or revised conclusions." Everyone can do some research under this definition.

Needed research in the stone industry can be divided in two parts: 1) production and 2) research as to use of products. Some of the production problems already mentioned should be restated to indicate possible lines of research.

Along another line, research should find some marketable use for the waste products which accumulate from stone operations. In the case of Consolidated Quarries, development of stone sand for use as fine aggregate has partly solved this problem. Now only -100 mesh is lost instead of $-1/8$ in. Acceptance of this product would be broadened through research on additives to overcome the natural harshness of broken stone.

The first step in finding profitable uses for fines that are not wasted is to gain thorough knowledge of the mineralogical, physical, and chemical properties of this material, then to try to match these properties of this material with possible uses, old or new. Some further treatment will generally be indicated, and there must be research on this as well as on the uses. The end result of such a program may very well be that desirable outcome—diversification of products.



Mining and Milling Cobalt Ore

Edwin B. Douglas

THE Blackbird mine is owned by Calera Mining Co., subsidiary of Howe Sound Co. The mine is located in the Salmon National Forest, Lemhi County, Idaho, about 22 miles west of Salmon and 220 miles north of Pocatello.

The thought of producing cobalt from a prospect that offered excellent opportunities stimulated everyone connected with this project. But as time went on, it was found that every phase of the development presented unusual difficulties.

The location was remote, in an area not served by railroads, a very mountainous terrain with dirt roads upon which the Forest Service did minor maintenance only in the summer. There were long cold winters with snow to fight five or six months of the year. It was a problem to haul in supplies and difficult to establish a camp and find personnel willing to rough it during a time of great prosperity in the country as a whole. As progress was made, the problem of mining flat-dipping, irregular, weak-walled, and weak orebodies had to be solved, and most perplexing of all, there was the problem of separating valuable cobaltite from worthless iron minerals. Fortunately these difficulties have not been insurmountable.

In bringing this property to its present stage of development, Calera Mining Co. has erected a modern townsite with facilities comparable to a small city housing about 900 residents. An industrial plant of the usual mine type is necessary for operation of the 1000-tpd mill.

Early History

Almost every productive district has behind it a series of starts and stops, attempts and failures. The Blackbird is no exception. Ore discoveries in the Blackbird district were first reported in 1893. The district evidently was active until 1907, with many small holdings prospected, and the Blackbird Copper-Gold Mining Co. consolidated the holdings and much of the ground now being mined. Plans were made to erect a small smelter for copper-gold ore, and thousands of cords of wood were cut and stacked in rows. The project was never carried through, but the cord-wood remains as a monument to the optimism of pioneers of the Blackbird district.

However, the district was not doomed to die. In 1915 Elwood Haynes, one of the founders of America's automobile industry, became interested in a source of cobalt, and through his efforts the Haynes-Stellite Co. acquired ground east of the Blackbird Copper-Gold Mining Co. ground. Between 1917 and 1920 Haynes-Stellite Co. mined and milled about 4000 tons of material. This enterprise likewise was unsuccessful.

After the Haynes-Stellite Co. ceased operations, there was little activity in the district until 1938, when the patented holdings of the Blackbird Copper-Gold Mining Co. were sold at a tax sale by the county. The new property owners, James and Howard Sims, were successful in interesting Uncle Sam Mining Co., which reopened two old tunnels and erected a 75-ton flotation mill. Like its predecessors, the Uncle Sam Mining Co. could not establish a profitable enterprise, partly because no attempt was made to recover cobalt.

Howe Sound Co., operating through its subsidiary, Calera Mining Co., became interested in the district in 1943. Just prior to this time the U. S. Government moved into the district in search of a domestic source of cobalt. With the help of the mapping and drilling by the U. S. Government and the Calera Mining Co. sufficient mineral was found to justify further underground exploration. This work was not feasible during World War II because of the manpower shortage, but at the close of fighting in Europe a contract was let for driving the Chicago adit. Since July

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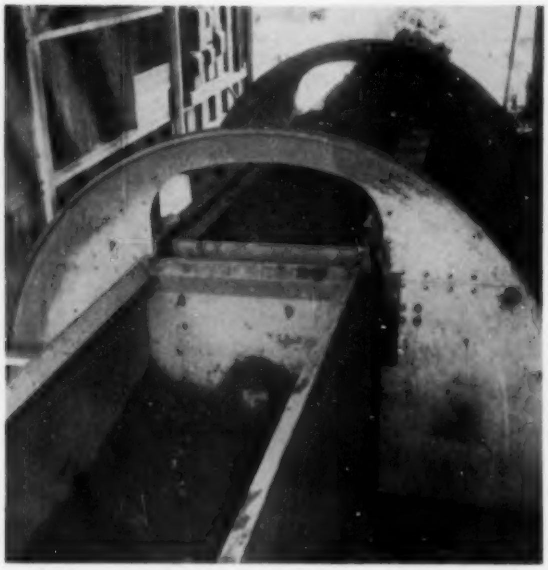


The dumping procedure used at Calera is designed to loosen muck from the sides and bottom of the cars.

1945 Calera has been active in the district. By 1949 sufficient ore had been blocked out to warrant full-scale development, and construction was started in that year.

Geology

The rocks of the Blackbird district are quartzites and metamorphased sediments of Pre-Cambrian age. The sediments strike northwest-southeast and dip 30° to 40° north. The sedimentary rocks are intruded by many basic dikes and by at least one large acid dike. The stratigraphy is not known, but the rocks are grouped according to type.

The district is bounded on the west by relatively firm quartzites; on the north by the granitic rocks of the Idaho batholith; on the east by well bedded quartzites cut by faults and breccia zones, many of which have been healed by silica and fine grained tourmaline; and on the south by argellite.

In the central section, an area about 2 miles wide by 6 miles long, the predominant rock is schistose quartzite, about 10 pct of which is biotite mica, causing the schistosity either along shear zones or along bedding planes. An irregular area to the north 1½ miles wide by 4 miles long contains garnet and chloritoid, but the rocks are essentially the same as those in the central part of the district.

Natural outcrops occur only at very scattered points because of the deep overburden and the thick forest vegetation. Geochemical prospecting, self-potential surveys, and exploration by bulldozer trenches have therefore been the means of locating favorable zones for diamond drilling and underground work.

Known commercial orebodies occur as hydrothermal replacements of predominately shistose rocks along shear zones. Three principal directions of shearing have been noted: 1) northwest shearing with moderate dips, 2) north-south shearing with steep dips, and 3) northeast shearing with nearly vertical dips.

Present underground workings have been driven northwestward along the general strike of the ore enclosing rocks for a total strike distance of 7000 ft. Veins and zones of mineralization have been disclosed at many places along this entire length. Several narrow ore shoots have been intercepted and two sizable orebodies have been indicated, the Chicago and the Brown Bear.

The Chicago zone has been exposed on four levels, the 6850, 7000, 7100, and 7200. Together with diamond drillholes, these levels have indicated the ore zone to be 1800 ft along the strikes and at least 350 ft deep. Widths range from 5 ft to as much as 60 ft and average about 15 ft. The ore zone consists of a group of pods and lenses striking about N30° W and dipping 50° to 75° NE. Intervening rock between ore shoots is generally mineralized but too low in grade to constitute ore under present conditions. The footwall of the zone is fairly well marked by a zone of strong shearing; the hanging wall is generally irregular. The Chicago ore, of high sulfide content, consists of an unusually vuggy pyrite, pyrrohotite and ore minerals, chalcopyrite, and cobaltite. Nonsulfide minerals include quartz, siderite, calcite, and very minor amounts of vivianite and ludlamite.

The Brown Bear ore zone has been opened up on six levels between the 7400 and 6850 levels for a vertical distance of 550 ft and a strike length of 1000 ft. This zone has features distinctly different from the Chicago. Massive sulfides are not the rule but occur in only a few spots and the cobalt in particular is finely disseminated in the schist. The sulfide minerals are cobaltite, chalcopyrite, and pyrite. Pyrrhotite, which constitutes much of the sulfide mineral in the Chicago, has not been observed in the Brown Bear. The zone is characterized by a series of mineralized schistose zones that strike within a few degrees of north, dip nearly vertical, and occur more or less on echelon toward the northwest. Their southern ends, at places merging with one another, appear to form continuous ore, while the northern ends of the pods extend as much as 100 ft and are up to 50 ft wide. Very indefinite walls are typical in the stopes and mining is controlled by sludge holes drilled into the walls.

Mining Methods

The basic method used at the Blackbird mine is a horizontal cut-and-fill, with hydraulicly placed sand slime for fill material. After the chute and manway have been constructed on the haulage drift, a small 4x4-ft untimbered raise is driven to the level above. This raise, a passageway for the sand-fill pipelines, is also used for stope ventilation, secondary access and, in some cases, service from the level above by

Because of the many different shapes these orebodies assume no set mining method can be used.

the use of a timber skip running on top of the ladders. The first slice of the stope is started, leaving a 15-ft pillar between its floor and the back of the haulage drift. This method has proved more successful than taking the first slice at the drift elevation.

In mining an 8 to 10-ft slice is breasted down on the sand fill by pusher-type drills in a series of rounds from 5 to 8 ft deep. No flooring is laid on top of the sand fill and each round is slushed to the chute by slushers of 10 or 15 hp and two or three drums, depending on the length and width of the stope. To minimize ore dilution with the sand fill, about a foot of broken ore is left on top of the sand until the final clean-up of the slice. The slice is mined 50 ft in each direction from the chute, about the most economical limit for these slushers. After the ore in the slice has been cleaned out, the chute and manway are built up to the same height as the top of the next sand fill. Chute and manway are constructed of 3x12-in. cribbing interlocked on the corners to allow ½-in. cracks between plates. A 4-in. divider separates the chute from the manway inside this cribbing and each compartment is 4x5 ft on the inside. The entire cribbing is wrapped with burlap and sealed at the bottom into the sand of the previous fill. Sand pulp of about 60 pct solids is then run into the stope from the surface storage tanks through 4-in. lines to the top of the stope raise, where the line is reduced to 3 in. In the stope the sand is distributed over the area through three 2-in. pipelines. The fill is placed to within 2 ft of the back. This permits removal of the 2-in. distribution sandlines and gives breaking room for the next slice being breasted down. Water from the sand fill drains out of the fill through the burlap around the cribbing. Mining can usually be resumed within 24 hr after the stope is filled.

The cut-and-fill system has proved very successful in the Chicago orebody, where the ore is fairly continuous and resembles a normal vein-type deposit. In this zone the chutes can run up the dip of the orebodies, and in most cases the ground is strong enough to stand unsupported until the fill is placed. Timber has been required around faulted areas and in the top slices of some stopes as they approach the oxidized zone near the surface.

In the Brown Bear ore zone the cut-and-fill system met with only partial success. Here, where the ore lies in flat plunging pods, chutes had to be started in the footwall of the pod and ended in the hanging wall in order to be steep enough to allow the ore to run. Due to the schistose character of the rock, the ground does not stand well, and it was found that after two or three slices were taken from a stope, even with close sand filling, the ground was so weakened that timber was required. As yet very little rock pressure has appeared, but the ground caves in large blocks caused by the intersections of faults and slips above the back.

Because of the many different shapes these orebodies assume, with additional complications caused by faulting, it appears that no set method can be used in mining this orebody. Mining of each individual stope must be solved as an independent problem.

The first stopes mined, using square sets, had been started as cut-and-fill stopes and converted to timbered stopes. This resulted in a large irregular horizontal slice supported by square set timber. Slushing was often a problem due to the irregular shape of the orebody, as it was impossible to get one slushing lane through the stope. Many times two additional cross slushes were required to move the ore to the chute. Also, this large horizontal slice required the timber to support a large area of ground, making it difficult to mine more than two floors vertically without filling. The latest system in the wider areas is to mine narrow slots three sets wide and to run from foot to hanging wall in stope length. This system will require more chutes but will keep a smaller horizontal area opened, eliminate the dog-leg slushing problem, and permit greater vertical distances to be mined before filling.

Because filling of the stopes is required for ground support, much thought and preliminary investigation went into the filling method. It was finally decided to use a deslimed product from the mill.

Mill tailings at about 30 pct solids are put through a 24-in. cone and thickened to 45 pct solids. The deslimed tailings are pumped from the mill sand tanks through a 6-in. woodstave line to the 7400 level sand tanks located above the Brown Bear shaft. Here the sands are thickened to about 60 pct solids either by decantation or through a second cone, depending upon the pouring schedule underground. The thickened sand product enters the mine either through a 6-in. vertical pipeline down the Brown Bear shaft or through an inclined line along the surface, depending on location of the stopes being filled. Distribution lines along the various levels to the stopes are 4-in. diam. Sands have been transported by this system a horizontal distance of 2800 ft with only 230-ft difference in elevation.

Because the mine is developed by adits, all above mill sand-tank elevation, quite an engineering problem was involved and some features of the installation are of interest. This problem, in its simplest terms, was to pump the 45 pct solids product about 1 mile with a difference in elevation of some 700 ft. This is accomplished by means of 11 Hydroseal slurry pumps divided between six stations, with no intermediate sumps.

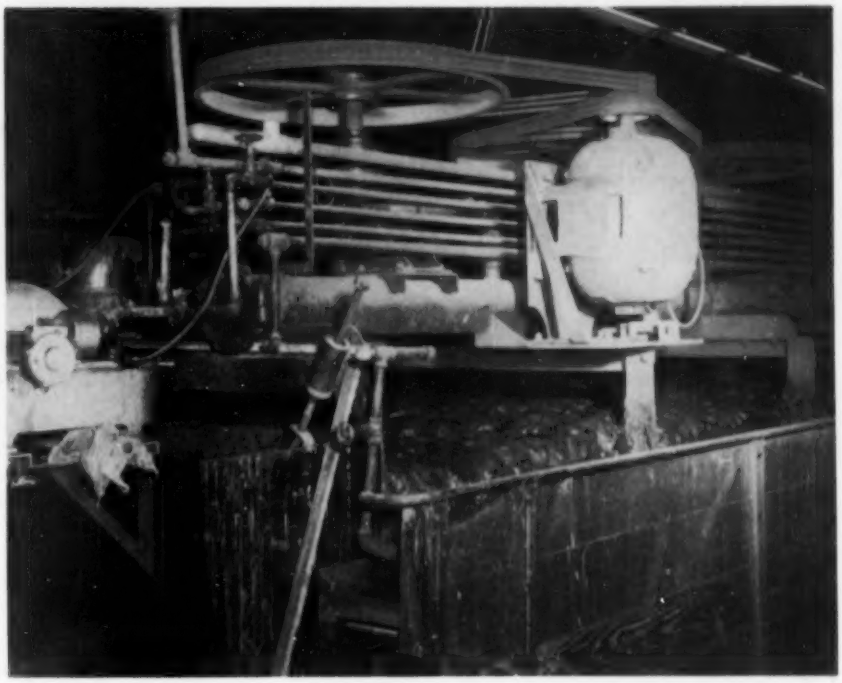
As far as is known, this system has more pumps in series than any other sand fill system. The method of starting the pumps in successive stages is unusual. An interlocked timing system is used whereby the maximum head is developed on each stage before the next series of pumps starts. This unique starting method allows for difference in distance and head between stations and also for large changes in pulp density. A system of electrically controlled dump valves is interlocked with the pump system to empty the line immediately in the event of power failure.

Milling

Several years ago, in a paper presented to the Idaho Mining Association, the opening remark on milling was: "The development of a suitable ore dressing treatment to separate from the ore a copper and cobalt concentrate has presented unusual difficulties." There was never a truer statement made regarding a milling process.

From the first it has been possible to make a satisfactory separation of the copper minerals from the cobalt and pyrite mineral. The separation of the cobalt from the iron minerals has been most tricky. Chemical reagents that depress iron minerals also depress cobalt minerals and, conversely, reagents that activate cobalt minerals also activate iron minerals. Gravity methods are not effective.

The entire milling problem has been further complicated by the requirements of the cobalt refinery. A chemical refining process has been used, the success of which depends considerably on a correct



Unusual difficulties were solved during the development of a suitable ore dressing treatment to separate copper and cobalt concentrates.

ratio of iron, arsenic, and sulfur in the concentrate. The refinery requires a cobalt concentrate grade of close to 17.5 pct. There is a problem, therefore, of separating just the right amount of iron from the cobalt. Early research work indicated a possible solution, namely, a partial roast of the iron-cobalt minerals to depress the iron. However, the key to present separation has been long conditioning of the pulp at elevated temperatures with lime. Even the conditioning is critical, as the cobalt and iron minerals can be over-depressed and the resulting float very poor. It has been found that the percentage of cobalt in the copper concentrate is a criterion of the effectiveness of conditioning.

After the pulp has been conditioned for approximately 1 hr at about 100°F and at a pH of 10.4, the cobaltite and pyrite are depressed and the chalcopyrite is readily floated using any one of several copper collectors. The copper tail becomes the iron section head where the scalping of sufficient iron is undertaken to give the proper iron-cobalt ratio for the concentrate. After the iron is scalped off, the cobalt mineral is activated by sodium sulfide and sulfuric acid is added in sufficient quantity to drop the pH to 3.2. At this pH the cobalt mineral floats readily with a cobalt tailing of 0.04 to 0.06 pct.

Even the mill water supply and tailings disposal has presented unusual problems.

To date there has been no success in using reclaimed water for the differential floats, and insufficient water exists at the mine elevation for milling. The dense forest mantle belies the amount of water that might be expected in the creeks. During the winter available water in the mine area drops to less than 200 gpm. To augment this supply a 6-in. woodstave line was brought from the West Fork of Blackbird Creek. The combined head of approximately 1400 ft was divided between four pump stations with equipment capacity for pumping 300 gpm.

Because the Blackbird mine is located in an area of steep-walled valleys and a general area where fine salmon, steelhead, and trout fishing attract many sportsmen, it was realized from the start that tailings disposal would be a problem. An acceptable location was found about 3 miles from the mill where the West Fork of Blackbird Canyon widens to perhaps 100 yd at its confluence with Blackbird Creek. Tailings are impounded behind an earth fill dam across the mouth of this canyon. A 6-in. concrete line 18,000 ft long was laid on water-grade down Blackbird Creek to the tailings site.

Uses of Cobalt

It is only recently that the uses of cobalt have expanded greatly. It finds many uses as an alloy because of its property of retaining hardness at relatively high temperatures (1000°C). High temperature alloys are now being used in the jet engines for airplanes, in gas turbines, and in high-speed tool steel. The miner uses cobalt when drilling with tungsten carbide bits where cobalt metal is the bonding agent for the tungsten carbide. A very important use is in permanent magnets, the most prominent of which is known as Alnico V, used in the magnetic separation of taconite and other ores and also in the loud speakers of radios and TV sets, electronic and electrical control equipment.

Cobalt is vital in lacquers, varnishes and paints as a drier. The oxide is used principally as the binder to metal for fine enameled surfaces.

Cobalt has minor uses in medicine and is a catalyst in hydrogenation of petroleum products.

Acknowledgments

The writer expresses appreciation to H. H. Sharp, president, Calera Mining Co., and V. H. Clarke and H. A. Pearse, vice presidents, for the opportunity to present this paper. He also thanks the members of the Calera Mining Co. for their assistance.

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Airborne Geophysics in the Search For Uranium in the Black Hills

Technique, procedure and methods to evaluate radioactive readings from airborne survey.

by Ted M. Rizzi

GEOPHYSICAL methods of discovering new mineral deposits have long been routine with both the petroleum and mining industries. Experience has shown that most subsurface structures and mineral deposits can be located, provided that detectable differences in physical properties exist. The main properties exhibited by the more common rocks and formations are: density, magnetism, electricity, electrical conductivity, and radioactivity. This entails five major geophysical methods: gravitational, magnetic, seismic, electrical, and radiation.

Applications of radioactivity measurements in geophysical exploration are concerned with location of concentrations of radioactive rocks and materials, such as uranium. The search for uranium ores and minerals has been carried on for more than 50 years, ever since the first atomic breakdown from uranium to radium was discovered. However, the comprehensive search now being made thoughout the world for radioactive materials is a direct result of the first atomic explosion and the subsequent development of new and more sensitive portable equipment to detect radiation.

Important developments in uranium exploration have taken place within the past few years. At the time of the first atomic explosion in 1945, the main source of domestic uranium was western Colorado, where uranium has been mined since 1899. As of 1945, known reserves were limited largely to the Morrison formation of western Colorado and a few other localities such as Temple Mountain, Utah. During the past nine years, uranium ore has been found in many new areas and ore has been mined from more than a dozen geologic formations ranging in geologic age from Pre-Cambrian to Eocene.

Most of these discoveries would not have been possible without aerial photography and use of the airborne magnetometer. Recently the airplane has come into use for radiation detection. In all instances where aircraft can assist in exploration, their chief function is to save time and manpower.

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The Homestake Mining Co. entered the field of airborne exploration for uranium in the spring of 1952, and since that time it has carried on a systematic search for uranium ores—mainly in the northern Black Hills of South Dakota and Wyoming—with the result that several small commerical deposits have been located.

Fortunately for all engaged in the search for uranium there is at least one consistent physical property of the substance which is always present—the property of radioactivity. Uranium ore on the ground can be located from the air by detecting the radiation given off by the naturally occurring element. Uranium ore is radioactive because it disintegrates into a series of different radioactive elements. This process of radioactive decay, omitting detectable radiations, serves as a guide to the location of any concentration of uranium. Three forms of radiation emanate from the naturally occurring element: 1) alpha particles, which have a range of only a few inches in air; 2) beta particles, which may range up to 8 or 10 ft in air; and 3) gamma rays, which may be detected up to several hundred feet through air, or even a 1000 ft or more from a particularly strong source.

The gamma radiation, therefore, is the one type of radiation of particular value in aerial prospecting for radioactive materials because gamma rays are the only ones that can be detected and measured at reasonable distance from the source. The gamma radiations are electromagnetic waves of extremely short wave length, similar to X-rays.

The production of radiation is not unique to uranium ore. There is radiation present at any location on the earth, and it is not detection of radiation only, but the increase of radiation that is important. The indication of natural radiation by an instrument is called the background reading. All indications of additional radiation, or anomalies, are measured in terms of increase over the background reading. Aerial prospecting for uranium ore, therefore, involves the finding of anomalous areas, that is, areas of gamma radiation that are higher than the general background level of radiation for the area under in-

Partial view of Black Hills, S. D., showing typical forms along outer edge of the hills. Flight lines are flown parallel to mesa tops or either sides of drainage in order to intersect any radioactivity which may be present.

vestigation. The background level of radiation is caused by cosmic rays, gamma radiations associated with cosmic rays, and the weak gamma radiations from the small amounts of radioactive material present in practically all materials. This background level of radiation varies from place to place but in general has a low average value.

In determining the presence of gamma radiation, the scintillation counter is several hundred times more sensitive than the Geiger counter and therefore best suited to airborne work. At Homestake the Model 118 Royal Scintillator has been used. This instrument employs a 2-in. sodium iodide crystal optically coupled to a photomultiplier tube, a minimum time constant of 1 sec, and a meter control switch that can be used to select any one of five meter ranges from 0.01 to 5 mr per hr. A full-scale deflection on the meter expresses a radiation intensity to the value selected by the meter control switch, that is, if the meter control switch is turned to the 0.01 mr per hr position, then a full scale deflection means that the instrument is being exposed to a radiation intensity of 0.01 mr per hr. Similarly, if the control switch is moved to the 5 mr per hr position, then full scale deflection means that the instrument is exposed to a radiation intensity of 5 mr per hr.

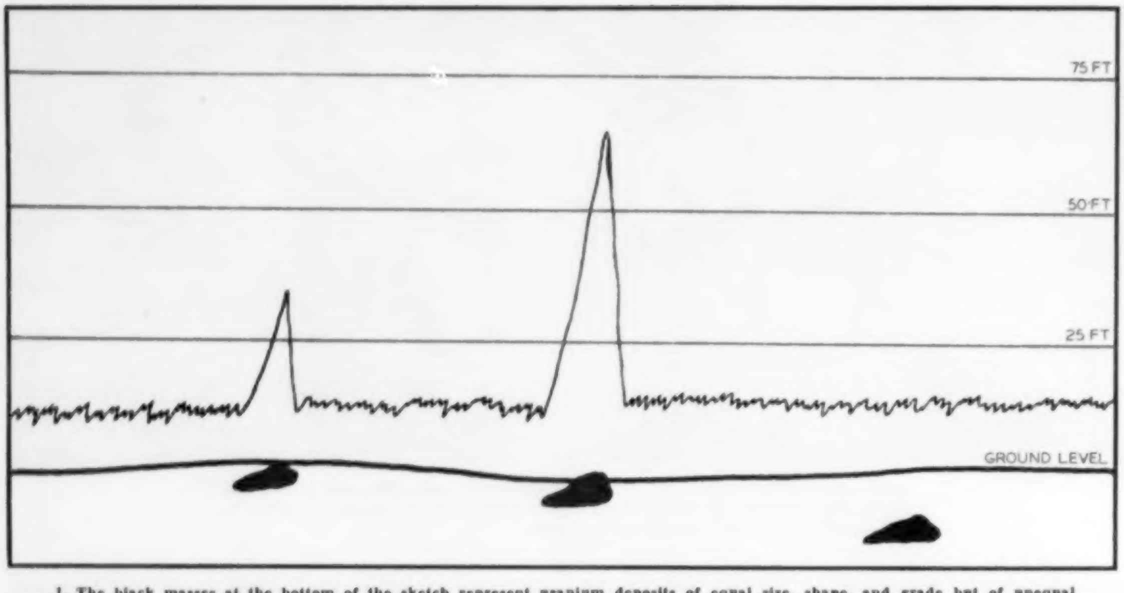
Because of the increased sensitivity to the gamma radiations from a radioactive source, the scintillation counter is also more sensitive to the cosmic rays and the gamma rays associated with the cosmic rays which form the background reading of any instrument. Normally the background value registered on the Royal Scintillator is from 0.005 to 0.02 mr per hr, or from one half to two thirds scale deflection on the third scale, with sudden cosmic ray bursts causing the needle to give momentarily higher readings. There is also considerable fluctuation of the needle because of the fast time constant required while in flight, and at times it is difficult for the observer to interpret these high readings without spending some time reflying the area to determine whether the high reading was actually due to radiations received from the ground or from some other source.

In aerial prospecting it is desirable to be able to use the most sensitive scale of the instrument and also to be able to take readings on the lower half of that scale. The reason for this is obvious. For a given radiation intensity, the needle swing will be much greater on one scale than on another. It is therefore much easier to notice a relatively large movement of the needle on the most sensitive scale than it would be to notice a smaller movement of the needle on a less sensitive scale.

To offset the effects of the background level of radiation the scintillator has been fitted with a lead shield. The greater mass of lead is near the sensitive portion of the instrument, reducing greatly the cosmic ray effect and giving a meter reading of about one quarter the reading without the shield. The lead shield enables Homestake to use the most sensitive scale of the instrument in aerial work. Some cosmic radiations are still being counted even when the shield is used, but the instrument now is recording mainly the weak gamma radiations from the small amounts of radioactive materials present in the ground below, and the needle movement is fairly steady. The lead shield makes the instrument directional, but the shield is arranged so that the sensitive part of the instrument or probe can be raised or lowered within it. This makes it possible to limit the angle of detection of the probe to a narrow one or to arrange it so that it can detect over a wide area. The probe is raised or lowered within the shield to fit topographic conditions as they arise.

Definition of the amount of radioactivity constituting an anomaly is still a matter of much discussion. It has been generally considered that twice the background count, or more, constitutes an anomaly. However, of the numerous anomalies of about twice the background investigated by Homestake during the past four years, it appears that anomalies of that order of intensity are usually unimportant, and only anomalies about four times the background prove out to be of any real interest. It is impossible to be definite in this matter, however, because a small anomaly may be due to the altitude and speed at which the aircraft is being flown, to the irregular terrain features which may well mask out any anomalies, and to the size and type of deposits. To be on the safe side, Homestake investigates on the ground all anomalies of at least twice the background that could be repeated in the air.

Because magnitude of response of the counter is a function of the altitude, the effect of the distance of



1. The black masses at the bottom of the sketch represent uranium deposits of equal size, shape, and grade but of unequal outcrop exposures. The deposit at the left has a smaller outcrop area than does the middle deposit, and the deposit at the right is buried under several feet of overburden.
2. The irregular wavy line above the deposits is an isorad line or line of equal radiation intensity. High points over the first two deposits are due to radiations being given off by the deposits. No high spot or anomaly is shown over the third deposit, since the radiations are blanketed out by the overburden before reaching the surface.
3. An instrument being flown at a particular altitude will be recording a certain background reading and the indications of any additional radiations will depend upon the altitude at which the instrument passes over a radioactive source and the amount of time it takes to pass over the source.

the counter from the radioactive source is extremely important. Actually, the intensity of radiation from a point source, that is, a source that is small in size in relation to its distance from the counter, follows the same inverse square law that applies to light: intensity varies as the square of the distance from the source. Loss of intensity due to air absorption must also be taken into consideration.

As previously mentioned, the instrument Homestake has been using in airborne work has a 1-sec time lag in its fastest position. This means that it takes the instrument 1 sec to receive and record the true radiation intensity from a radioactive source for the altitude at which it is being flown. An airplane, for example, flying at 60 mph will be traveling at 88 fps, and in all probability the airplane may already have passed over a small deposit before its presence is indicated by the instrument, and the signal may be a very weak one. So the magnitude of response of the instrument to radiations from a radioactive source is not only a function of the altitude but also of the amount of time during which the counter is being exposed to the radiations from the radioactive source.

Airborne prospecting can result in location of outcropping uranium ore only if the outcrop is large enough or of high enough grade. With the aircraft flying at some particular altitude there may not be enough uranium ore outcropping at the surface to supply radiation for a reasonable increase in background reading. The most effective airborne prospecting is done with small highly maneuverable aircraft flying close to the outcrop area. The closer it is possible to approach the outcrop the more anomalies will be detected, and for airborne prospecting for radioactive materials it is also desirable to fly as slowly as is safe.

To refer to the diagram, for example, an instrument flown at 50 ft will record a certain background level of radiation for that altitude. In passing over the first deposit no increase over the background reading will be obtained because the anomaly given off by the deposit is a weak one due to its small outcrop area.

In passing over the second deposit, the instrument will show an increase over the background reading because it is cutting through the isorad line. Whether the increase in reading will be sufficient for the observer to note as an anomaly will depend upon the amount of increase over the average background reading the instrument has been recording. A very slight deflection of the needle would probably pass unnoticed or would be noted but eliminated as a possible anomaly because it might not have been twice the background reading.

If the width of the anomaly curve at the point it is cut by the instrument is approximately the same in flying time as is the time constant for the instrument, then a true reading will be recorded by the instrument for that altitude. If the airplane carrying the instrument is flying at 60 mph and the instrument has a time lag of 1 sec, then the instrument will record a true reading for the radiations from the radioactive source if the spot the instrument cuts through the anomaly is 1 sec flying time in width. In this case there would be a good sharp deflection of the needle. If the width of the anomaly curve is appreciably less than 1 sec flying time across, then the needle deflection would be considerably less.

At an altitude of 25 ft, the possibility of detecting the anomaly over the first deposit would again depend on the conditions just mentioned. A definite anomaly would be recorded over the middle deposit, regardless of air speed, because of the closeness to the large outcrop area.

Flight experiments over known point deposits at various altitudes and air speeds have shown that there is an optimum speed and elevation at which

a particular deposit can be detected, again depending on the size and grade of deposit. Ideally, in order not to miss small occurrences of potential value, a survey aircraft should be flown as close to outcrop as is safe and should maintain a constant altitude above the ground to eliminate the troublesome psuedo-anomalies resulting from topographical expressions of high readings with decrease in altitude. Because it is impossible to do this over the rugged terrain of the Black Hills, however, the observer must at all times estimate his height above the ground and interpret his meter readings accordingly.

The amount of coverage of the area that will be scanned on the ground from a certain altitude can be estimated to be a path twice as wide as the altitude at which the aircraft is flying. At an altitude of 50 ft, therefore, a path on the ground approximately 100 ft wide would be within range of detectability. It must be remembered, however, that flying in a deep canyon or gorge, the instrument sees more surface and a resulting increase in reading is obtained which the observer must interpret correctly.

Before an aerial exploration program is begun, it must be decided where flight is to begin and what geologic horizons are to be explored. Prospecting for uranium should be most successful if carried out in areas where: 1) uranium has already been found or 2) geologic conditions are similar to those existing where uranium has been found.

When prospecting new country it is essential to keep in mind the type of rock uranium seems to prefer. Much rock with favorable features is not mineralized, but most ore at present is being mined from light or gray sandstone, deposited by streams, which contains carbonized matter or is interfingered or interbedded with shale or mudstone. Since most of these features are not recognized by the observer while in flight, this information is obtained through a study of geologic maps and literature prior to flying the area.

Since uranium ore was discovered in the Lakota-Dakota sandstone in the southern Black Hills, near Edgemont, Homestake has confined most of its flying program to these two sandstone horizons. These two formations are of varying thickness around the Black Hills and are separated by a shale member (Fuson) 30 ft to 190 ft thick. The Lakota sandstone forms a prominent part of the Hogback Ridge that extends around the Black Hills; the Dakota sandstone crops out in a narrow zone along the outer slope of the Hogback Ridge. Homestake has also spot checked many of the other geologic formations that outcrop in and around the Black Hills, but to date no appreciable amounts of uranium have been found. A new horizon was recently opened up to airborne prospecting when promising levels of radioactivity were discovered in various lignite beds in northwestern South Dakota and southwestern North Dakota.

An aerial exploration program consists of two general types of flight operation, grid flying and rim flying. A grid pattern is flown over the area in which a detailed radiation intensity or isorad map is desired, but it is limited to areas of low relief on a near horizontal plane. Rim flying, used by most airborne prospectors, consists in flying as close to the ground or canyon and valley walls as possible to intersect any radioactivity. This type of prospecting, which must be done by a skillful pilot, requires a counter wih high sensitivity and a slow-flying plane.

Aerial exploration at Homestake has been rim flying, necessitated by the irregular terrain features of the Black Hills. Flying has been carried out in a Super-Cub type of plane of 115 and 135 hp, and flying is done from just after sunrise to about 10 a.m. It has been found that wind conditions are generally more calm during the early morning hours, the air is cool and thus more stable, and flying for the day is accomplished before the sun has had a chance to heat the air and produce the thermals which result in much turbulence over the irregular terrain later in the day.

An attempt is made to follow the rim exposures at a fixed distance, which varies according to the topography, from the possibly mineralized horizon. It is not always possible to follow all re-entrant canyons in this manner, but a large part of the exposed formation can be followed maintaining visually near-constant altitude above the terrain.

In flying over hundreds of miles of various geologic formations, Homestake prospectors have gained considerable experience in using the scintillator. An air speed of 60 mph is maintained, using half flap, and flights are made as close as possible to the objective, which may be 25 to 100 ft distant depending on topography. Contour flying in the Hills, rather than straight line flying, has proved more satisfactory for maintaining constant altitude above ground.

Using rim flying entirely, it has been possible to locate several mineralized areas later proved to contain mineable uranium. High anomalies were also obtained which proved to contain no uranium or thorium. In these cases radioactivity is believed due to residual radium.

Constant reference to geology of the area being flown is necessary for intelligent use of the scintillator. A sudden doubling of the background, if sustained, probably means the crossing of a contact into an area of more acid rocks of higher potassium content. Shales have a higher radioactivity than other types of rock surfaces. Granites and Tertiary intrusives will give a momentarily higher reading due to the presence of radioactive elements associated with them, but these rocks carry too little uranium or thorium to be of commercial interest at present. Although susceptibility of the scintillator to changes in potassium content of the rocks will record abrupt changes in readings, generally the causes are apparent. The scintillator will record the presence of gamma radiation from whatever the source in the vicinity, and the observer must interpret the cause.

Whenever a high radiation zone is encountered, the particular area is reflown in several different directions to enable the observer to pinpoint the apparent source of radiation. Anomalies thus found are noted on aerial photographs or spotted as accurately as possible on maps if photographs are not available for the area being flown. Guided by aerial photographs—the best method of pin pointing anomalies—field parties sent to an area to ground check an anomaly can go to the exact spot.

In conclusion, it should be understood that airborne prospecting is applicable to deposits that crop out at the surface or are very near to the surface. Also, it should be understood that the scintillation counter is not offered as an instrument that will locate uranium deposits but only as an instrument that will provide valuable clues as to their possible location. Airborne prospecting can only be considered in its positive aspect—failure to discover mineralization does not eliminate an area from investigation by other methods.

Design and Evaluation —

Turbine Mixers in Metallurgical Applications

by N. H. Parker, G. Gutzeit, and J. G. Papailias

IN its many forms, the turbine has greater latitude of application than any other mixing device. It is readily adaptable to all tank shapes, from the shallow *pan* to the tall and narrow *stovepipe*. Although turbine mixers can be adapted to most tank conformations, experience has shown that there is a preferred shape for optimum efficiency in each service. In the accompanying drawings this relationship is illustrated by the shallower tank for slurry suspension. The taller tank is preferable for continuous staged systems, or maximum retention of the gas phase in absorption.

No single turbine covers the entire range of these applications with maximum efficiency. The back-sloped blade offers maximum power economy. The straight blade develops a higher discharge velocity for the dispersion type of service requiring localized shear. The same high shear environment is provided by the back-sloped impeller with stator ring assembly, which permits the most positive discharge control, Fig. 1. Positive stream control is critical in solids suspensions.

It is possible to adjust any mixer element, whether paddle, propellers, or some form of turbine, to satisfy almost any service. When this is done, however, sacrifices are made in horsepower consumption, initial capital expenditure, maintenance costs, and low mixing efficiency.

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Using the Selection Chart: The selection chart, Fig. 2, separates mixing operations into five primary and four secondary services. Many operations combine two or more. When the service has been defined, the most suitable mixing device can be selected, and the criteria are established. In the determination of specific shape relationships from the last three columns of the selection chart, the following factors are of prime importance in solids suspension operations found in metallurgical applications: *percent solids, particle size, specific gravity of solids, settling rate of solids, and gravity difference between solids and liquor.* Acting severally and together, these factors affect the proper operation of the mixer, and can, if neglected or improperly used, cause unnecessary expenditures due to power or mixer failure.

To illustrate, assume the leaching of a 50 pct slurry of an ore of 3.8 sp gr ground to 20 mesh in a dilute acid, with heat of solution to be removed by cooling coils. The primary requirement is suspension of the solids to promote the reaction, then removal of the heat generated.

The criteria for solid suspension are circulation and liquid velocity sufficient to overcome the settling rate of the solids. The same criteria also apply to good heat transfer and reaction. The large particle size and gravity difference between solids and solution suggests fast settling. The best impeller position is therefore on the vessel bottom, so that the radial discharge across the bottom will lift solids up into the tank. To maintain maximum distribution of solids, yet allow sufficient depth of

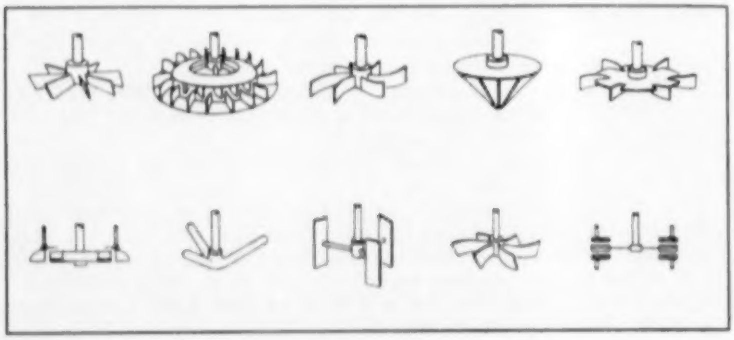

Fig. 1—Left to right, top, turbines and dates of introduction: pitched, 1910; shrouded with stator, 1913; open tilted, 1932; cone lifter, 1932; center disk, 1934. Bottom turbines are: gas disperser, 1935; three-blade, 1935; Brumagin, 1937; variangle, 1938; vane-disk gas, 1939.

SELECTION CHART				SHAPE RELATIONSHIPS FOR TURBINE DESIGNS		
SERVICE	MIXING DEVICE	RANGE	CRITERIA	TANK DIAMETER TO IMPELLER DIA RATIO	TANK HEIGHT TO DIAMETER RATIO	IMPELLERS AND POSITION
BLENDING	TURBINE		1. VOLUME CIRCULATION	3:1 TO 6:1	UNLIMITED	SINGLE OR MULTIPLE
	PROPELLER					
	PADDLE					
	TANK VOL.	→ 1,000,000 GALS				
DISPERSION (IMMISCIBLE SYSTEMS)	TURBINE		1. DROP SIZE CONTROL 2. RE-CIRCULATION	3.0:1 TO 3.5:1	1:1 1:2 IN STAGED MIXERS	AT/OR BELOW CENTER LINE OF LIQUID CHARGE
	PROPELLER					
	PADDLE					
	FLOW	→ 1,000 GALS/MIN				
REACTIONS IN SOLUTION (MISCIBLE SYSTEMS)	TURBINE		1. INTENSITY 2. VOLUME CIRCULATION	2.5:1 TO 3.5:1	1:1 TO 3:1	SINGLE OR MULTIPLE
	PROPELLER					
	PADDLE					
	CHARGE VOL	→ 20,000 GALS				
DISSOLUTION	TURBINE		1. SHEAR 2. VOLUME CIRCULATION	1.6:1 TO 3.2:1	1:2 TO 2:1	AT/OR BELOW CENTER LINE OF LIQUID CHARGE
	PROPELLER					
	PADDLE					
	CHARGE VOL	→ 10,000 GALS				
SOLIDS SUSPENSION	TURBINE		1. CIRCULATION 2. VELOCITY	2.0:1 TO 3.5:1	1:1 TO 1:2	DEPENDING ON PARTICLE SIZE. 1. IMP. DIAMETER OFF BOTTOM. 2. ON BOTTOM.
	PROPELLER					
	PADDLE					
	% SOLIDS	→ 100 %				
GAS APPLICATIONS	TURBINE		1. CONTROLLED SHEAR 2. CIRCULATION 3. HIGH VELOCITY	2.5:1 TO 4.0:1	4:1 TO 1:1	1. MULTIPLE - LOWEST ONE IMPELLER DIAMETER OFF BOTTOM. 2. SELF-INDUCE, JUST BELOW LIQUID LEVEL.
	PROPELLER					
	PADDLE					
	GAS VOL	→ 5,000 CU FT./MIN.				
HIGH VISCOSITY APPLICATIONS	TURBINE		1. VOLUME CIRCULATION 2. LOW VELOCITY	1.5:1 TO 2.5:1	1:2 TO 2:1	SINGLE OR MULTIPLE
	PROPELLER					
	PADDLE					
	VIS.	→ 1,000,000 CPS				
HEAT TRANSFER	TURBINE		1. VOLUME CIRCULATION 2. HIGH VELOCITY ACROSS TRANSFER SURFACE	RELATED TO OTHER SERVICES	DEPENDS ON OTHER SERVICES BEING PERFORMED	SINGLE OR MULTIPLE. IMPELLER OPPOSITE TRANSFER SURFACE WHEN USING COILS
	PROPELLER					
	PADDLE					
	CHARGE VOL	→ 20,000 GAL				
CRYSTALLIZATION OR PRECIPITATION	TURBINE		1. CIRCULATION 2. LOW VELOCITY 3. SHEAR CONTROL	2.0:1 TO 3.2:1	2:1 TO 1:1	SINGLE AT/OR BELOW CENTER LINE OF LIQUID CHARGE
	PROPELLER					
	PADDLE					
	CHARGE VOL	→ 20,000 GALS				

Fig. 2—Selection chart separates mixing operations into five primary and four secondary services. Many operations combine two or more.

liquid for the cooling coils, the maximum tank height ratio of 1:1 from the chart would be used.

The ratio of tank diameter to impeller diameter, usually referred to as *impeller ratio*, is regulated by reaction and suspension, with the latter controlling because of particle size. Tank depth and particle size suggest a large impeller diameter, or a ratio of about 2.5:1. As the circulation pattern now established is radially across the bottom, and up the sides, slurry will flow across and through a helical coil for good transfer rate. This pattern will be assured by four full baffles mounted inside the cooling coils. The completed design is shown on the left in Fig. 3.

In the previous example, because the solids were ground 20 mesh and because the ore gravity was 3.8 and the slurry 50 pct solids, the impeller position recommended was the simplex, located on the tank bottom. If the same ore were ground 90 pct −200 mesh, the impeller position could be changed to two thirds of the impeller diameter off the bottom of the tank. Because of the difference in particle size, the impeller position and the tank shape can be changed and still maintain optimum operation.

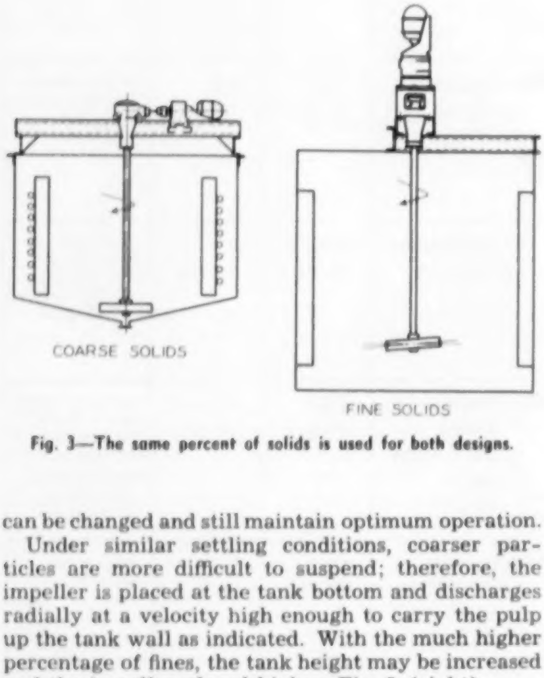

Fig. 3—The same percent of solids is used for both designs.

Under similar settling conditions, coarser particles are more difficult to suspend; therefore, the impeller is placed at the tank bottom and discharges radially at a velocity high enough to carry the pulp up the tank wall as indicated. With the much higher percentage of fines, the tank height may be increased and the impeller placed higher, Fig. 3 (right).

Fig. 4 shows a design used for pulps of fines consisting of about 60 to 75 pct solids, presently installed for wet storage in a sinter plant. Even though the solids in this case have a rapid settling rate, *hindered settling* assists the mixer in keeping the solids suspended.

Other Useful Design Charts: Four graphs aid completion of preliminary designs: Fig. 5, horsepower of back-sloped impellers; Fig. 6, corrections for peripheral speed; Fig. 7, circulation; Fig. 8, curves for retention time for continuous staged systems.

Two illustrations will show how these graphs can be used. Assume a design has been worked out from the Selection Chart, Fig. 2, for a mixer to repulp filter cake that is 80 pct −100 mesh. The impeller is to be 54 in. diam. The relationship of horsepower to impeller diameter, shown in Fig. 5, is based on the use of open back-sloped impellers of the same ratio of diameter to blade width, and the horsepower is given for a constant peripheral speed of 700 fpm, commonly called *normal speed* for a turbine impeller. Power given is based on water, of 1.0 sp gr.

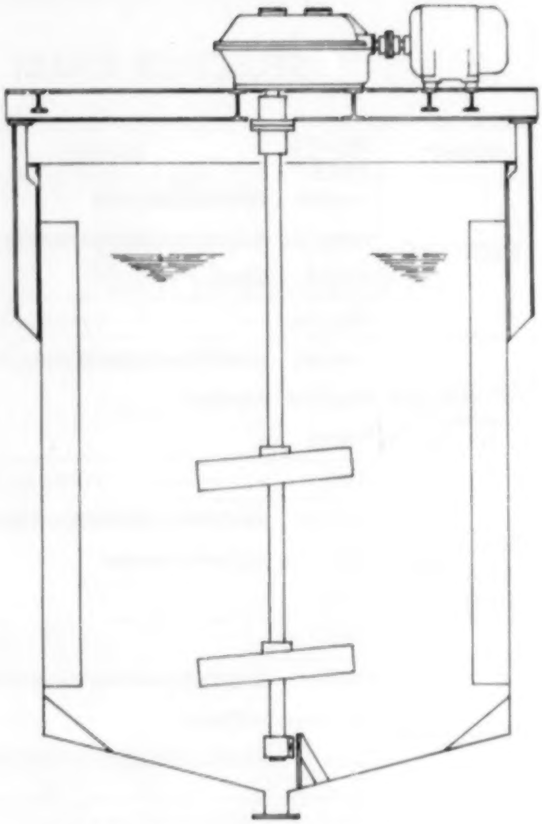

Fig. 4—Design used for high solids pulps of fines.

Fig. 5 shows that the impeller would draw 8 hp on water at normal speed. It is now necessary to know the relationship between revolutions per minute and the peripheral speed at 700 fpm. The approximate relationship

$$\text{Rpm} = \frac{2700}{\text{diam, in.}}$$

gives the shaft speed necessary to operate the impeller at normal speed. A slide rule calculation indicates that 50 rpm is the required shaft speed.

The general formula $HP = KD^5N^3\rho$ relates the power, HP, to the diameter, d, of the impeller, the speed, N, in revolutions per minute, and the specific gravity, ρ, of the material being circulated. The constant, K, is a function of the Reynolds number and of the mixing element used. These have been included in the construction of the HP graph, with the specific gravity taken as that of water, 1.0 sp gr. Since specific gravity in the general formula is a direct multiplier, when the HP graph is used for slurries or fluids of specific gravity different from 1.0, the horsepower taken from the graph is multiplied by the specific gravity of the material being mixed.

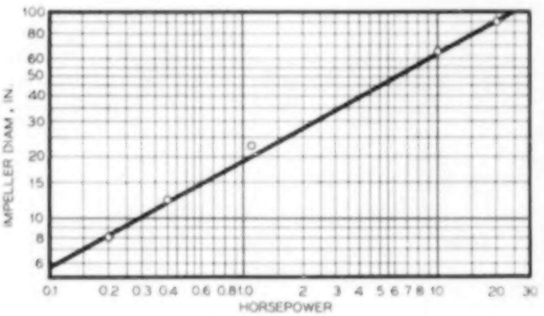

Fig. 5—Horsepower required by efficient one-impeller turbine mixers of various sizes at 700 fpm peripheral speed.

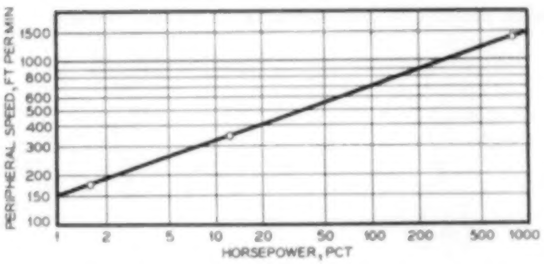

Fig. 6—This chart is used for corrections of peripheral speed.

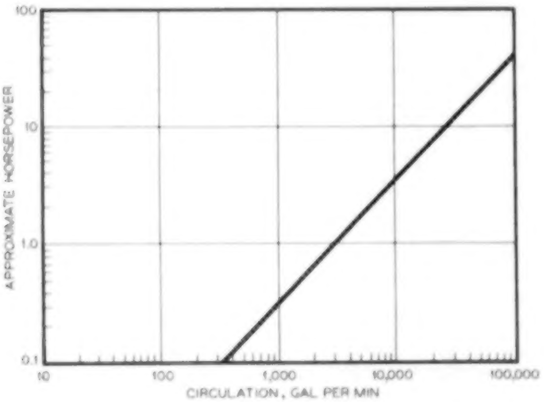

Fig. 7—Circulation achieved vs horsepower input required. Horsepower on water, data from Fig. 5.

Therefore, if the gravity of the repulped solids is 1.39, the horsepower required to handle the final slurry properly will be 8x1.39, or 11.1. Since the next higher standard motor horsepower available is 15, ample safety factor is provided for drive loss, for any variation in solids density affecting the final specific gravity, or for periodic load variation.

However, if the impeller is going at a peripheral speed more or less than 700 fpm, Fig. 6 gives the required correction factor, which is also a direct multiplier.

Once the required horsepower is established, a third graph (Fig. 7) can be used relating available pumping capacity or circulation to horsepower input. Circulation capacity of the impeller or impellers, divided by tank volume, gives a practical measure of the intensity of the mixing provided. This number is tank turnovers per minute, and the higher the number, the greater the intensity.

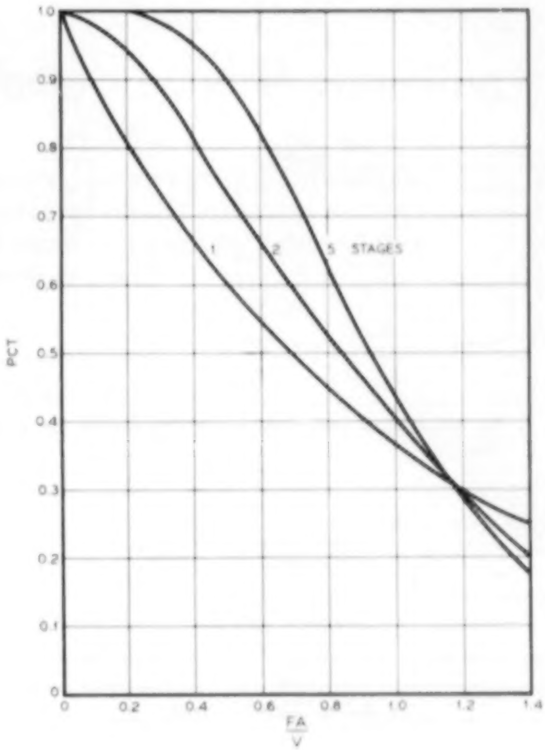

Fig. 8—Holding time in continuous staged mixers. Graph after McMullin and Weber.

To illustrate the use of the curves of Macmullin and Weber for retention time in continuous staged mixers, see Fig. 8, a good example would be the leaching circuit at the Nicaro plant of the Nickel Processing Corp. The aerators used in the existing plant and those to be installed in the new expansion are shown in Fig. 9. The operation consists of dispersing air in the slurry made up of the quenched roasted ore suspended in an ammonium carbonate solution. The problem was to determine the number of stages of mixing required to provide the retention time necessary for maximum extraction by solution of nickel from the ore.

In pilot plant work on the process, it was realized that retention time was a critical factor and that when the best cycle was established, it would be necessary to extrapolate the leach circuit design to achieve the necessary retention time on a 3000-tpd plant. Data taken on the pilot plant Turbo Aerators were successfully extrapolated to the full plant by use of the curves in Fig. 8.

The ordinate represents that portion of the material that will be retained in the system for the assured holding or retention time. The abcissa is a number to which is equated, F in gallons per minute multiplied by the assured holding time, A, divided by the total volume of the system, V. From this relationship, if two of the variables are known and the number of stages selected, the total volume of the system, the assured retention time, or even the flow rate could be found. With the information developed as to the tank shape from the selection chart, a complete preliminary design can be established for any continuous flow process.

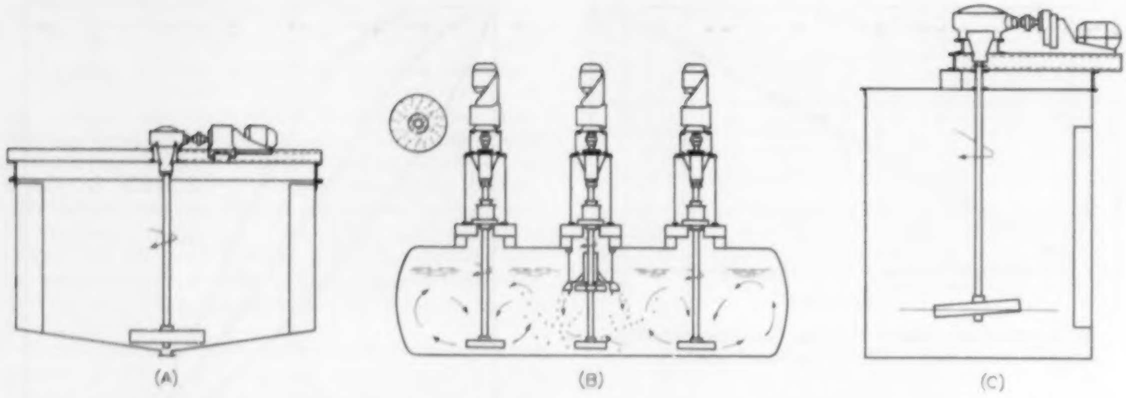

Fig. 10—Right, A, solids suspension at Vitro Uranium Co. B, reduction autoclave, Whitaker Metals Co. C, leaching, Tennessee Corp.

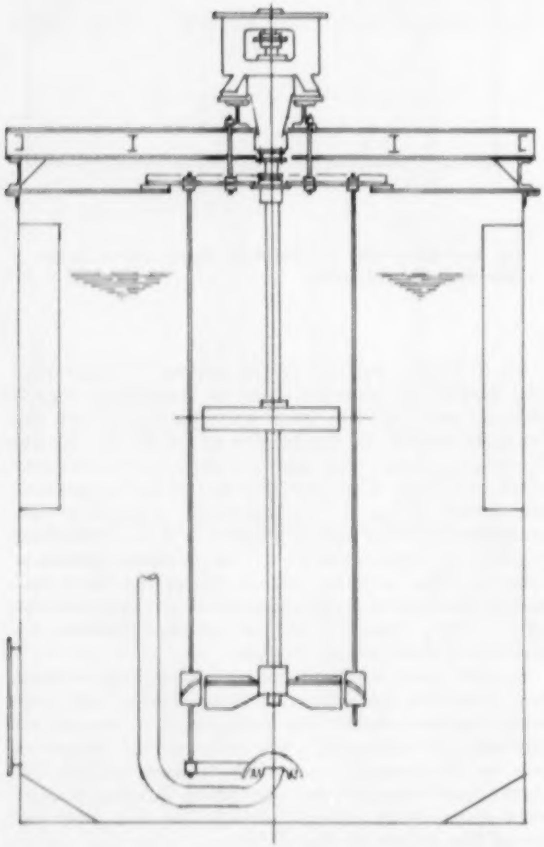

Fig. 9—Aerators of this type are used in the existing plant and will be installed in the new expansion at Nicaro, Cuba.

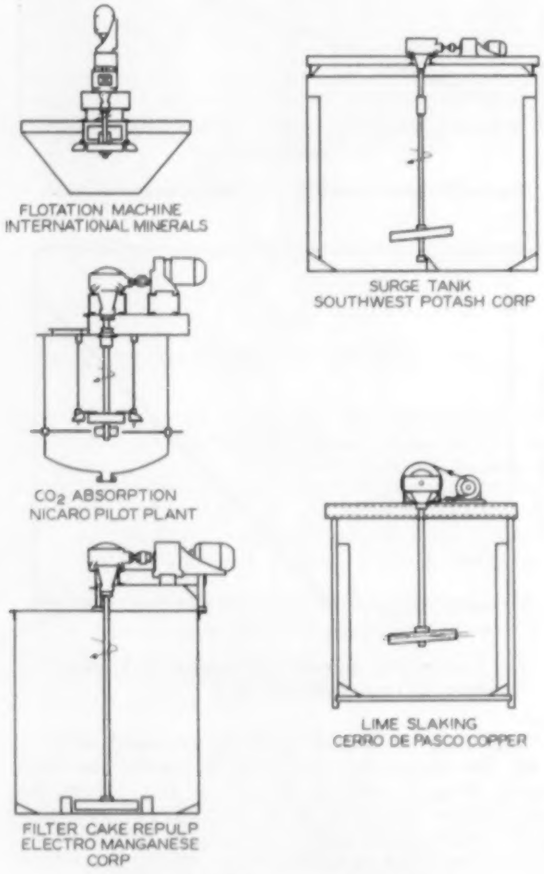

Fig. 11—Typical metallurgical applications.

To refer again to Fig. 9, and the Selection Chart, the tank height to diameter ratio can range between 1:1 and 4:1, for aeration and gas absorption operations. A long gas path is desired. The Turbo Aerators at Nicaro are provided with a Gas Absorber impeller and Down Flow ring for optimum gas distribution and a downward directed flow to give the gas a longer path in the slurry. In the Nicaro design the ratio of height to diameter is 1.2:1. The upper impeller recirculates the aerated pulp in the upper zone of the tank.

The turbine mixer is not limited to the several applications discussed, as can be seen by the wide variety of services listed in the selection chart. Some of the other applications, Figs. 10 and 11, typical of extractive metallurgy include flotation machines, filter cake repulping, surge tank agitation, reagent mixing, and CO_2 absorption.

The Gravity Meter in Underground Prospecting

For the past six years gravity meter surveys have been used for underground prospecting. Evaluation of results indicates definite potentialities under proper conditions. Equipment, procedures, and methods of handling data are described.

by William Allen, Jr.

FOR the past six years gravity surveys have been used for underground prospecting in the copper mines at Bisbee, Ariz.

The primary purpose of the surveys has been to reduce the diamond drilling and crosscutting necessary for exploration. Since many of the orebodies are small, and geologic control is not always apparent, any information that will direct the drilling and crosscutting is highly desirable.

Because of extensive development and exploration work in the copper mines at Bisbee, it has been possible to cover more than 630,000 ft of crosscuts on 30 levels with the gravity surveys. In the process the gravity procedures have been refined to a high degree.

Density Contrast: For a gravity survey to be successful, a sufficient density contrast must exist between the geologic feature sought and surrounding host rocks. Most mineralized areas will provide this contrast if fairly massive bodies are present.

In the Bisbee area the entire sequence of formations, except for alluvium, appears to have specific gravities ranging from 2.65 to 2.70. These values have been determined by means of a large number of cut samples and diamond drill cores. As a further check, vertical gravity differences have been used where nonmineralized sections are known to occur.[1]

The only known major gravity disturbances result from mineralization that has increased the density and the voids that have decreased density. The voids are caused by mining operations and by underground water movement that has developed several areas of caverns.

Equipment: While not absolutely essential, a small rugged gravity meter, such as the Worden meter, is highly desirable. A tall tripod, about the height of a transit tripod, permits instrument set-ups in deep water and in locations where fallen timber and muck piles make it impossible to use a short tripod. An additional advantage of a tall tripod is that it places the meter in the center of the crosscut, reducing the error caused by the crosscut void.

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Discussion of this paper, TP 4212L, may be sent (2 copies) to AIME before May 31, 1956. Manuscript, April 24, 1955. Chicago Meeting, February 1955.

Size and weight are important, since the only satisfactory means of operating the meter underground is to carry it by hand. A backpack can be used in rare instances but is usually a hindrance because of the close station spacing. The operator's ability to move through tight clearances will improve survey coverage, as it is then possible to move through raises and caved areas and to pass mine cars and machinery with a minimum of trouble.

Station Control: Gravity stations are normally located every 100 ft along the crosscuts, at each intersection, and in the face of all stub crosscuts. In areas of high gravity relief, or where small anomalies might be expected, stations may be located at 25 or 50-ft intervals. When possible, the stations should be offset to avoid effects of raises or other voids. The gravity stations on a level are tied to one or more base stations, which are usually located at the shaft or near the portal of an adit.

The base stations may be part of a gravity control net that extends to each level in the mine as well as to the surface. Such a net extending throughout the potential area of the surveys is highly desirable, as it is then possible to compare all gravity stations on a uniform basis. The stations that are part of the base net should be carefully established by multiple readings and, if necessary, by a least squares adjustment of the loops.

In some instances where levels do not have a shaft station, or where access may be blocked by caving, it may be necessary to establish secondary bases at the top and bottom of the raises that are between levels.

Under fair conditions 70 to 90 gravity stations can be located and run in 6 hr by a two-man crew. The best field procedures depend on conditions.

Reduction of Field Data: Most of the time required to produce a final gravity map is consumed in processing the data. Each meter reading must be corrected for a minimum of five factors that affect the gravity value in addition to the density contrast being sought. These factors are 1) instrumental drift, 2) station elevation, 3) topography, 4) latitude, and 5) regional gravity gradient.

Mine openings, such as stopes and raises, will affect the value. However, it is seldom practical to make corrections for these voids. Usually a rotation is made on the field note on the station, and any

The Transactions papers appearing in Mining Engineering during 1955 will subsequently appear in AIME Transactions Volume 205 and may be permanently referenced to that volume.

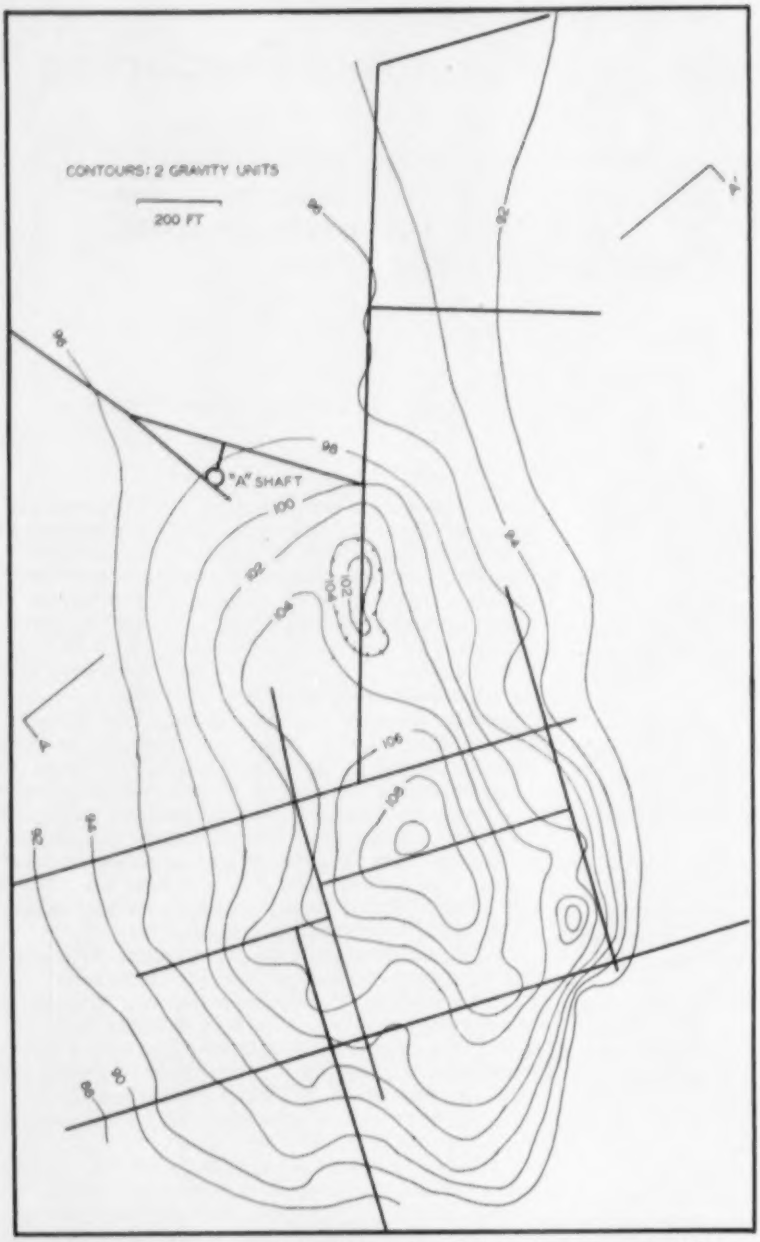

Fig. 1—A portion of a typical underground residual gravity contour map, 10th level. For clarity the contour interval has been increased and stations and shorter crosscuts omitted. The approximate center of the mineralization on this level falls within the 108 contour. The negative closure near the center of the map is caused by a small elongated body above the level. A small mass of sulfides on and directly below the level causes the small closure at the lower right.

anomaly is considered with the extraneous effects in mind.

The drift correction is a simple linear correction to account for variations in the base readings. Except for minor tidal changes these variations are caused by mechanical changes in the meter system.

A major problem in any gravity survey is to obtain the elevation of each station. In a well surveyed mine it is practical to use the elevation of the track at each location in lieu of actually running levels. It will be necessary to interpolate elevations between the points that are actually known, but if grades have been maintained, this presents no special problem. It is usually not necessary to correct for the actual height of the gravity meter above the track, since only a minor error is introduced in the final results. This error is essentially uniform for all stations.

Topographic correction is by far the most time-consuming and laborious part of the entire gravity survey. A greatly expanded version of Hammer's tables[1,2] is used for corrections, which are made for all topography out to a radius of 65,150 ft around each station. In practice it has not been found necessary to correct each gravity station directly. It is quite feasible to use a contour map of the topographic correction prepared from grid points at which the correction has been determined. In general, the deeper the level the more uniform the correction. Extreme care must be used in applying this method to shallow levels, especially if the topography has high relief immediately above the level. In some instances each station must be corrected.

To prepare an adequate topographic correction for an area, a good topographic map is essential. Most U.S. Geological Survey maps are satisfactory for the purpose, especially if they are enlarged four or five times. Lack of a good map often makes a satisfactory gravity survey impossible.

The latitude correction, which accounts for flattening of the earth at the poles, is generally uniform over small areas.[3]

The regional gravity correction is used primarily to eliminate any gradients occurring uniformly throughout the survey area. Removal of the gradient will accentuate the smaller anomalies and flatten nonanomalous sections.

The second-derivative methods used on the surface are not practical underground because of irregular station locations and lack of broad coverage.

There is definite indication that it is not always desirable to remove a regional gradient, which is often associated with district-wide structures and is therefore of interest.

Presentation of Data: Final presentation of the gravity data can be handled in three ways. First, and least useful, is the gravity profile. This method

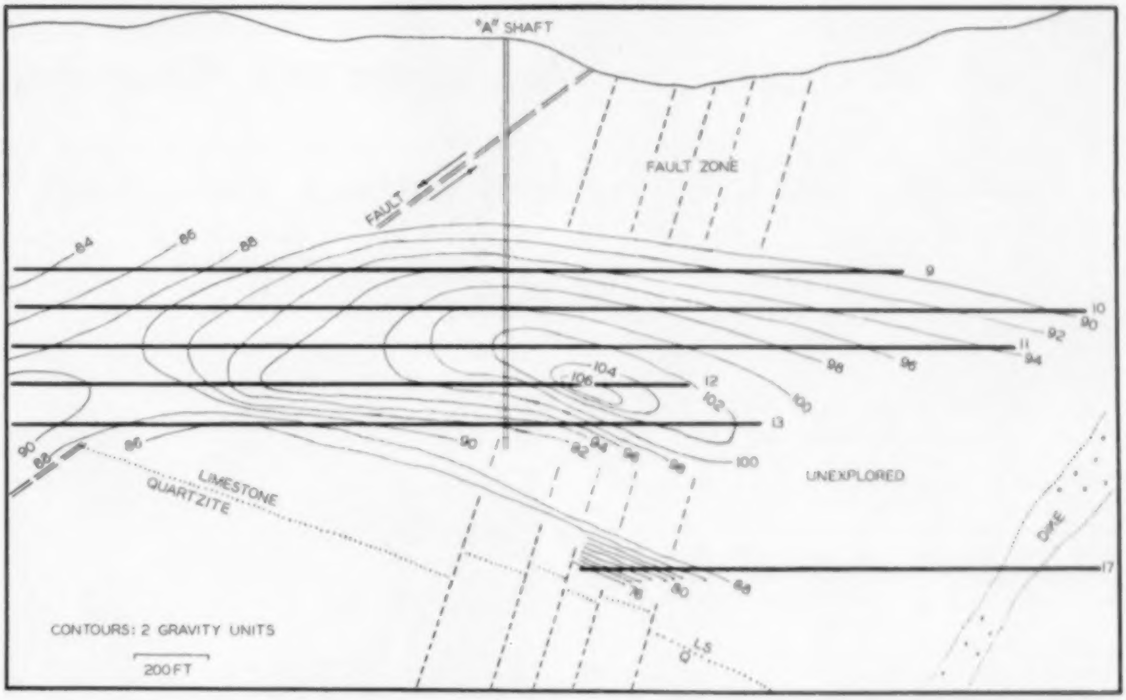

Fig. 2—Gravity section along AA' in Fig. 1. The top of the mineralization is approximately between levels 11 and 12. The rake is to the right and into the figure. Lower and lateral limits are not established.

is best used in long crosscuts where very little gravity control is available to either side.

The second method is the gravity contour map as shown in Fig. 1. The contour interval that has been found best for underground work is 0.1 milligal or 1 gravity unit. Contouring is sometimes difficult because of a lack of proper control, but most anomalies show in one form or another. There has been some question as to the advisability of extending gravity contours over distances up to 500 ft or more between control points; however, experience indicates that very little error occurs, especially after there is some knowledge of the gravity trend in an area. Small anomalies will be missed in such areas.

The third method of presentation is presently feasible only in underground work. This consists of a gravity contour section through the mine area, Fig. 2. Used in conjunction with the contour map, the gravity section makes available a three-dimensional gravity map of the mine. It is quite feasible and often very informative to prepare a model of the mine with gravity contours superimposed on the levels. While there is not sufficient information for full evaluation, it appears possible to project trends of mineralization as well as general upper and lower limits. This would obviously be of great value.

When a borehole gravity meter is developed, the gravity section should be easily adaptable to both petroleum exploration and mining exploration in which large diameter holes are used.[3]

Interpretation: A major advantage of the gravity survey over other geophysical methods in underground work is that any anomaly can be attributed to only two factors—an increase or decrease in mass. This may be contrasted to other methods where anomalies are often caused by some factor totally unrelated to mineralization.

There are two types of gravity anomalies underground, the gravity high and the gravity low.

The high can be attributed either to a center of mass below the level or to a lack of mass above the level. If densities are uniform, the mass below the level must of necessity be caused by mineralization, while a lack of mass above the level may be due to stoping, gossen zones, or caverns.

The gravity low is the converse of the gravity high.

It is absolutely essential that any gravity interpretation be closely correlated with known geology and known stoping if correct evaluation is to be made.

Disadvantages: The major disadvantage of underground gravity surveys is poor lateral control. It is easy to miss a small sulfide mass located within 20 or 30 ft of a crosscut, if it is on or near the level of the survey. When it is possible to prepare gravity sections, lateral control can often be improved.

Many gravity anomalies cannot be resolved. In most cases this is due to insufficient geologic information. In such instances, if the anomaly appears to warrant it, recommendations must be based on the geophysical information alone.

Conclusions: Results of the underground gravity surveys indicate that excellent data can be obtained where there are fairly uniform densities and adequate coverage. The chief advantages over other methods of geophysical prospecting are the greater ease in interpretation of data and the lack of interference from operations.

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Genesis of Titaniferous Magnetites and Associated Rocks of the Lake Sanford District, New York

by J. L. Gillson

THE big mass of anorthosite in the Lake Sanford district and the bodies of titaniferous magnetite that occur in a small area near the south margin of the mass have been described repeatedly, and the puzzling problems of the genesis of the rocks and ores have stimulated the deductive reasoning of geologists for a hundred years. Some of these men have taken a leading part[1-15] in describing the anorthosite rock and forming hypotheses concerning its origin; some have contributed most to the study of the ore deposits;[illegible] and some have made classical studies[illegible] introducing a basic approach to the problems of the anorthosite rock and genesis of the titaniferous magnetites.

Much of the writing on the problems of rock and ore genesis has been based on physical-chemical reasoning rather than on field and microscope observations of the rocks and ores. Most of these geologists conclude that the anorthosite was formed by true magmatic crystallization, involving previous settling or squeezing out of ferromagnesian minerals from the parent magma. The associated gabbro is a separate segregation or later intrusion, and the lighter colored and finer grained facies of the anorthosite is either a chilled border zone of the anorthosite or the result of granulation by crushing of the coarse, dark blue rock of the main mass. This blue rock, called the Marcy type, is named after the higest peak in the range. The finer grained and lighter colored rock is called the Whiteface, after another mountain appropriately named for a cliff colored by this facies of the anorthosite.

In explaining the genesis of the ores the authors have reached no general agreement, but all consider them to be dominantly the result of magmatic processes rather than the result of replacement by pneumatolytic or hydrothermal processes. However, in the conclusions of two authors, Osborne[illegible] and Stephenson,[illegible] the reader is told positively that the ore minerals were introduced after the wall rock was solid and also that the oxides are later than the silicates. This seems to mean that the ore came in as a later intrusion, like a dike, since Osborne speaks of filter pressing and later injection of a residual magma. Stephenson agrees in general with Osborne but disagrees with his conclusion that ore does not grade into country rock. Singewald[illegible] calls

Fig. 1—Quarry showing both faults. At left is post-ore that fault displaces; there is no alteration along fault zone. At right is pre-ore; the rock is intensively altered along the fault. The white is andesine that has replaced the dark labradorite.

in "mineralizers" to participate in magmatic deposition. Stephenson ends his paper with a statement that replacement was the dominant process for the introduction of ore with anorthosite. Hence both Singewald and Stephenson call on pneumatolytic or hydrothermal processes to supplement magmatic crystallization from a silicate melt, but they cannot quite bring themselves to state that the orebodies formed in that way. In their conclusions the ores are still magmatic segregations and magmatic injections and—almost as an afterthought—"replacements." According to Bateman[illegible] there was a gravitational accumulation process of the iron silicates and oxides, which were injected later. Ramberg[illegible] explains the ores entirely on physical-chemical theory, as does Evrard.[illegible] Buddington and associates,[illegible] in a paper presented to the Geological Society of America in 1953, believe the facts observed prove that the magnetite-ilmenite formed at magmatic temperatures.

The present writer's conclusions, given here, were first presented in 1947 in *Industrial Minerals and Rocks.*[illegible] Since the papers of Ramberg, Bateman, and Buddington, published after 1947, gave no recognition to those conclusions that differ from their own, the writer has taken an opportunity to restudy the area in the field, and to examine a number of microscopic sections, and now presents the evidence more thoroughly than he had an opportunity to do in the

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Discussion of this paper, TP 4175HI, may be sent (2 copies) to AIME before May 31, 1956. Manuscript, Nov. 1, 1954. Lake Placid Meeting, October 1954.

earlier paper. The accompanying illustrations (Figs. 1-16) are submitted as evidence for his conclusions.

Evidence for Pneumatolytic Replacement

Observations made by the writer are summarized under the following headings: 1) localization of ore zones, 2) structure, 3) mineralogy, and 4) paragenesis.

Localization of Ore Zones: The orebodies are confined to a very small area in the south end of the anorthosite massif. Although this fact may not be convincing as to the mode or origin, it does indicate that the process of ore deposition was local and specific and not a general one with widespread application.

Structure: Structure was studied by the writer only at the largest orebody, that at Sanford Lake, which is so well exposed in a large quarry because it has been in production since 1942. There are two basic features of the structure:

1) Zones are parallel, trending northeast-southwest. There is some evidence that the zones form a plunging syncline. The zones of anorthositic wall rock, anorthositic ore, gabbro, and gabbroic ore are at least partially repeated on an island in Sanford Lake. Banding or gneissic structure in the rocks is roughly parallel to this trend of the major zones and to the axis of the syncline.

2) Some faults, almost at right angles to these northeast-southwest zones, have displaced the zones of anorthositic ore, gabbro, and gabbroic ore. As shown in Fig. 1, the andesinization of the Marcy into a typical Whiteface anorthosite as later described is accentuated along one fault.

This structural zoning and alteration along fault lines has been and can be interpreted in various ways. There is no doubt that differential pressure was active at one time and that resulting gneissic banding controlled in position the later alteration of the original rock or rocks into these parallel zones. The fault obviously displaced solid rocks, and the fact that the mineralogical alteration followed the fault zone indicates that it had not resulted from differentiation of a liquid magma, but came about by metasomatism of a solid or almost solid rock.

Mineralogy: The minerals in the rock and ore are not exciting in number or variety. They are labradorite, andesine, hornblende (with brown to green

Fig. 2—Massive magnetite-ilmenite ore, retaining original gneissic character seen in much of the anorthosite.

Fig. 3—Marcy anorthosite with wedge-shaped mass of ore. A garnet reaction rim is found along the side of the ore. In the block at the left the Marcy anorthosite is crossed by a vein of andesine.

pleochroism), augite, hypersthene, biotite, garnet, scapolite, apatite, magnetite, ilmenite, spinel, and calcite.

In this list two minerals, garnet and scapolite, stand out, since they are seldom if ever found in rocks of simple pyromagmatic origin. To emphasize the magmatic origin of the minerals, calcite can be ignored and mentioned casually simply as a late mineral, possibly due to recent weathering, and the abundant apatite can be explained by the proponents of magmatic crystallization as having been filter-pressed out of the anorthositic magma along with the ferromagnesian and oxide minerals. The scapolite and garnet, however, never arrived in the rock by filter-pressing, for they had been formed after the rock was solid, as is shown convincingly in the photographs.

The scapolite, which occurs in the big feldspars, formed during andesinization of the feldspars and is obviously a replacement mineral. It is widespread in the Whiteface type of anorthosite but is not common in the gabbro or the ore. The shape of the crystals is shown in Fig. 16.

The garnet is found in many associations. Most conspicuously, in megascopic observations it occurs as random aggregates, some several inches across, although most are smaller. Garnet shows no evidence of crushing and formed after the differential movement was over. In thin sections many of the garnets are seen to be in contact with the big feldspars as *reaction rims*, explained by proponents of magmatic differentiation as resulting from reaction of the iron-rich magma, squeezed out of the parent magma, with the big labradorite crystals floating in the still liquid melt. The microscopic evidence, Figs. 9 and 15, favors the replacement interpretation.

A few characteristics of the several minerals should be noted. The big labradorite crystals owe their dark color to a cloud of minute inclusions. Most of the inclusions are too small for positive identification, but some are a green pyroxene. Many of the big feldspars have been altered by that process called *saussuritization*, which indicates a soaking of the rock by hot solutions, since the minerals of *saussurite* are epidote, mica, and chlorite. Most of the andesine grains, on the other hand, are clear and

Fig. 4—Gabbro with labradorite *phenocryst* just as seen both in Whiteface anorthosite and in ore.

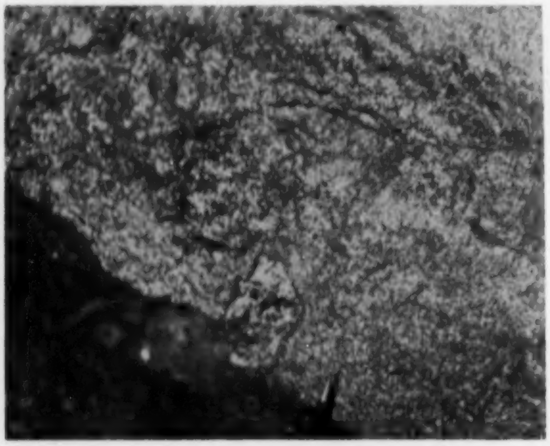

Fig. 5—Large labradorite crystal in massive ore. There is a garnet reaction rim between the feldspar and the ore.

Fig. 6—Whiteface anorthosite miles away from the orebody shows relic of large labradorite crystal in finer grained, fresh, nearly white andesine. (Picture supplied by J. G. Broughton.)

Fig. 7—Gabbro with cross-cutting band of ore with garnet reaction rim between ore and gabbro. There is a crystal of labradorite at the point of the pencil.

Fig. 8—Block of massive ore, with big crystal of labradorite remaining unreplaced.

fresh and generally allotriomorphic in shape. A few show strain in their extinction positions.

Many of the big feldspars are crossed by narrow veins of the more sodic variety. Some of these veins are big enough to see in place with the naked eye (Fig. 3). Others are seen only in thin sections.

The composition of the ferromagnesian minerals is of no special interest, but the crystal shapes are noteworthy. According to Bowen's classic sequence, the ferromagnesian minerals should have crystallized early in these rocks, taking euhedral or subhedral form, but they are obviously later than the feldspar. This conclusion of relative age by shape and boundary relationships is based on evidence presented 1) in earlier papers[illegible] by the writer in which he gave criteria for deuteric minerals, and 2) in papers by Newhouse,[illegible] who has published a long study describing replacement shapes and the diagnostic characteristics of replacement minerals.

In 75 to 90 pct of the observations made the magnetite and ilmenite, except in the richest anorthositic ore, are confined inside the ferromagnesian minerals (including garnet) and in 50 pct of the observations, inside the garnet. These oxides, according to Bowen, should have started crystallization very early. In the ore they are later than the garnet and hypersthene, as shown in Figs. 10 and 11. Since the garnet was late and formed by replacement of the feldspar, the oxides must also have formed by still later replacement in the solid rock.

Paragenesis: The writer believes that the original rock, the Marcy-type anorthosite, composed almost exclusively of large crystals of labradorite, was soaked with solution first rich in soda and as a result much of the original feldspar was replaced by andesine. This process he calls *andesinization*, which is similar to the familiar albitization of many granites. The Whiteface anorthosite was in turn converted to a metagabbro by the introduction of iron and magnesium, forming the ferromagnesian silicates. Both gabbro and anorthosite were converted to ore by the still later introduction of more iron and also titanium, which formed the metallic oxides. The most convincing evidence of this parentage of Whiteface anorthosite, gabbro, and ore is afforded by relics of the big labradorite feldspars found in all the other rocks and in both gabbroic and anorthositic ore, see Figs. 4-8. These old feldspar relics are in all stages of preservation, from crystals 2 in. across or larger to faint remnants now hardly more than an outline. They are exceedingly numerous in every rock type and can be found during the briefest examination of benches in the MacIntyre mine pit. It is inexplicable to this writer why they have not been mentioned in any previous report, as they prove that all the rock types and the ore had a common heritage in the Marcy anorthosite itself.

When the basic fact is recognized that each of the rock types has resulted from introduction of new minerals into the Marcy anorthosite, it is easy to work out the obvious paragenetic sequence of the minerals as shown by their boundary relations in thin sections. The andesine feldspar replaced the labradorite and probably the scapolite was formed simultaneously. This andesinization, which was very widespread, preceded the introduction of iron, magnesium, and titanium, possibly by a considerable time interval. It occurred through much of the whole massif, whereas later changes were more local.

When the iron and magnesium solutions started coming in they were confined to a much more re-

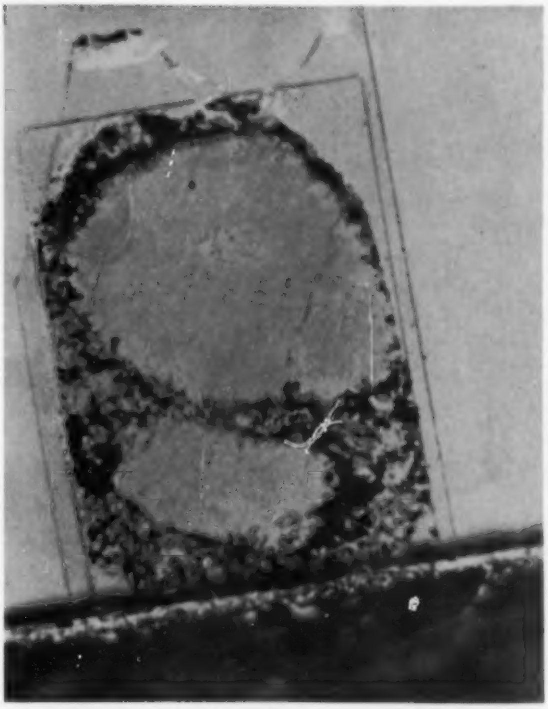

Fig. 9—Thin section of anorthositic ore, photographed without the use of a microscope. Two large crystals of labradorite are shown, surrounded by reaction rim of magnetite, garnet, and small crystals of andesine.

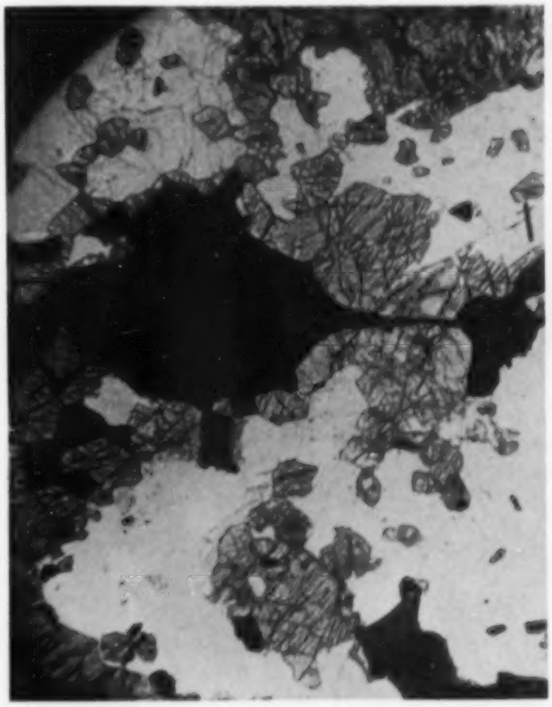

Fig. 10—Micrograph of gabbroic ore. Magnetite in hypersthene. The magnetite is obviously later than the silicate. Low magnification.

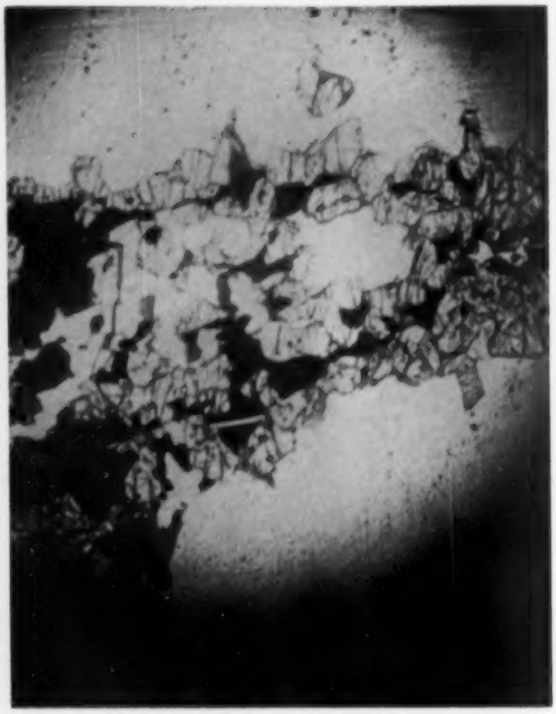

Fig. 11—Micrograph of Whiteface anorthosite. A section of a reaction run between two large labradorite grains. Note inclusions in feldspar which give it its dark color. Magnetite was formed later than silicates.

Fig. 12—Low grade anorthosite ore. Magnetite is found more frequently in garnet than in other silicates. Medium magnification.

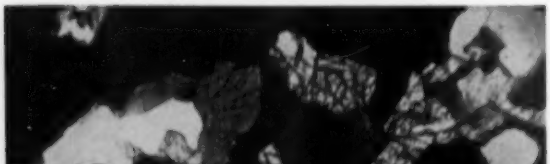

Fig. 13—Micrograph of typical gabbroic ore. Magnetite is formed later than the silicates.

Fig. 14—Magnetite reaction rim around a relic of a large feldspar. Low magnification.

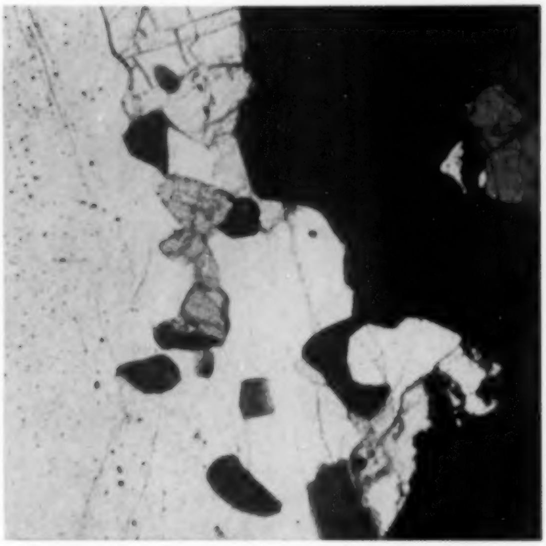

Fig. 15—Gabbro ore. Edge of reaction rim between ore and feldspar. Note inclusions in large feldspar which give the labradorite its color. Andesine grains in the reaction rim are clear.

stricted zone than the andesinization, and the solutions followed the gneissic, crushed, and faulted zones, yielding a banded zoning of gabbro and ore in small, specific localities.

These later solutions reacted with the older feldspar, with which they were not in chemical equilibrium, and formed the reaction rims so characteristic of the gabbro and gabbroic ore. The absence of crushing of these minerals, as well as the existence of the garnet, which is not a stress mineral, shows that the differential stress was over. Such banding as occurs is pseudomorphic of the original gneissic structure. Within the bands the micas and hornblendes are not in uniform arrangement as they are in gneisses and schists.

A great increase in iron, magnesium, titanium, and phosphorus in the solutions formed the magnetite, ilmenite, apatite, and spinel that are the ore minerals. These final solutions did not follow the same channels as those preceding them. Hence large masses of anorthositic ore formed where magnetite replaced Marcy or Whiteface anorthosite in which the iron-magnesium silicates had not formed previously. In the gabbro these oxides replaced the ferromagnesian minerals. As the gabbro is finer grained than the anorthosite, the gabbroic ore is finer grained than the anorthositic. For some reason not readily understood it is also richer in titanium than is the anorthositic ore. At the end of the mineralization, calcite was introduced.

Conclusions

The processes of granitization, albitization, and replacement of older rocks, particularly in the pre-Cambrian, is recognized and accepted by scores of petrologists. In this more basic rock, an anorthosite, the albitization became andesinization. Later soaking by iron-magnesium and titanium solutions formed a metagabbro and a titaniferous iron ore locally along crushed and faulted zones into which the solutions had access.

This study does not reach back to the problem of genesis of the Marcy anorthosite. The writer accepts

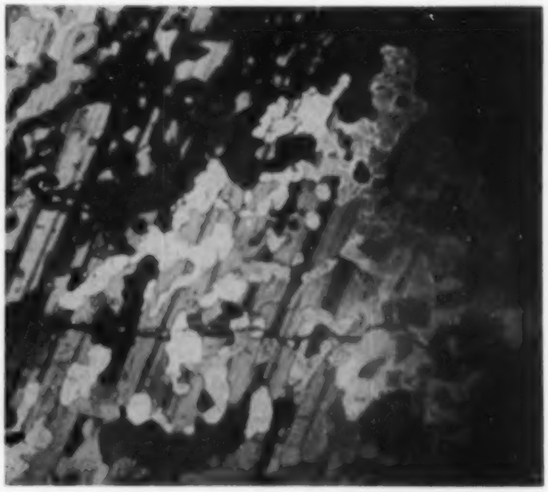

Fig. 16—Whiteface anorthosite. Feldspar partly replaced by scapolite. Crossed nicols, moderate magnification.

it as an already solid rock and advances no ideas of its origin.

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Ionic Size in Flotation Collection of Alkali Halides

by D. W. Fuerstenau and M. C. Fuerstenau

Studies of the collection of alkali and ammonium halides utilizing vacuum flotation techniques and contact angle measurements show that ionic size controls the flotation of these halides with amine salts as collector. Contact angles of air bubbles on sylvite in saturated brines were measured as a function of such variables as collector addition, length of collector chain, and pH of the brine. No contact occurs between halite and an air bubble in brines containing dodecylammonium acetate as collector.

LONG-CHAINED aliphatic amine salts have been used for the separation of sylvite (KCl) from halite (NaCl) by flotation.[1, 2] It is puzzling how these two minerals, which are so similar chemically and crystallographically, can be separated by this method. Gaudin[3] has postulated that the difference in floatability of halite and sylvite with salts of primary amines depends on ionic size:

> In the case of amine flotation, the cation would attach itself to the chloride. I have a speculation there, which I cannot prove, that the ammonium group, that is the $-NH_3$ group in the amine, floats potassium chloride because the dimensions of this group as it has been measured in other compounds is almost identically the dimensions of the potassium ion, quite different from the sodium ion, and so it fits where potassium had been, in place of it and not attached to it.

Apparently, because an aminium ion (RNH_3^+) is much larger than a sodium ion, it cannot fit into the lattice of halite. Taggart also has speculated that ionic size may control the floatability of sylvite.[4] The object of this experimental investigation has been to test this hypothesis and to study what controls the adsorption of cationic collectors at the surface of sylvite.

Since collection is to be approached from the viewpoint of ionic size, the ionic radii that are of interest in this work are presented in Table I. The values of the ionic radii of the ions listed in Table I, except NH_4^+, are those given by Pauling.[5] Several different values for the radius of the ammonium ion have been given, but that of Goldschmidt[6] seems to be preferred. The radius of the charged head of a dodecylammonium ion is assumed to be the same as that for the ammonium ion.

Little experimental work has been reported in the technical literature concerning the separation of sylvite from halite by flotation. Guyer and Perren studied the separation by flotation of 50 pct binary mixtures of NaCl, KCl, NH_4Cl, $NaNO_3$, KNO_3, K_2SO_4 and Na_2SO_4 using either oleic acid or a sodium sulfonate as collector.[7]

It is possible to measure floatability under actual flotation conditions where all three phases, air-water-mineral, are present by vacuum flotation tests[8] and contact angle measurements.[9] Both of these techniques were used in the experimental approach in this paper.

Experimental Method and Materials

The vacuum flotation tests were run with glass-stoppered pyrex graduated cylinders. Twenty-five ml graduates were used to test the floatability of all salts studied except rubidium and cesium salts. For each test distilled water containing the desired collector concentration was saturated with the salt to be floated. Sufficient salt (−48 mesh) was added to leave about 2 ml of solids in the bottom of the graduate. After the graduate had been agitated several minutes to saturate the solution with air, a vacuum was applied. If the salt were floatable in the collector solution, the gas bubbles attached themselves to the particles, and the particles floated to the surface. In determining the floatability of the expensive Rb and Cs halides, the experiments were run in 10 ml graduates with about 1½ ml of collector solution initially.

Contact angles were measured in the usual manner except that the solutions had to be previously saturated with the mineral to avoid dissolution of the crystal. Solutions for studying contact angles were made by adding the desired amount of collector to a saturated brine, giving the collector concentration in molarity. The mixture was agitated until dissolution of the collector was complete, with the exception of those concentrations greater than about millimolar. At these high concentrations complete dissolution of the collector was impossible. The face of the mineral to be tested was a freshly cleaved crystal of halite or sylvite. The mineral was placed in the brine and conditioned with collector for at least 15 min, which was found to be long enough to obtain a maximum value for the contact angle. The temperature remained constant during each experiment. The experiments were run at 24°C ± 2°C.

For contact angle measurements, a crystal of halite from Carlsbad, N. M., was used. Several samples of sylvite were used in this work: a crystal of sylvite from Stassfurt, Germany; a crystal from Carlsbad, N. M.; and a crystal of chemically pure potassium chloride. Saturated brines were made from reagent grade chemicals and distilled water.

Table I. Ionic Radii in Angstrom Units

Li^+	0.60	Mg^{++}	0.65	F^-	1.36
Na^+	0.95	Ca^{++}	0.99	Cl^-	1.81
K^+	1.33	Sr^{++}	1.13	Br^-	1.95
Rb^+	1.48	Ba^{++}	1.35	I^-	2.16
Cs^+	1.69				
NH_4^+	1.43				

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The vacuum flotation tests were made by D. W. Fuerstenau in the Richards Mineral Engineering Laboratories, Massachusetts Institute of Technology. The contact angle experiments were made by M. C. Fuerstenau at the South Dakota School of Mines and Technology.

Discussion of this paper, TP 4156B, may be sent (2 copies) to AIME before May 31, 1956. Manuscript, July 21, 1955. New York Meeting, February 1956.

For the vacuum flotation tests, reagent grade salts were used. The flotation collectors were prepared by Armour & Co., Chicago. In these experiments the amine salt was added either as the acetate or chloride, but in a saturated brine the collector should be considered as the chloride salt because of the tremendous predominance of chloride ions.

Experimental Results

Vacuum Flotation Tests: To study whether or not ionic size controls the floatability of halides with aminium ions as collector, the best procedure appeared to be to ascertain the floatability of all the readily available alkali and ammonium halides by means of vacuum flotation tests. In this manner it is possible to vary, over wide ranges, not only the size of the cations but also anions of substances that are nearly identical chemically. A series of vacuum flotation tests were made with each of the alkali and ammonium halides, except RbF and CsF, in the manner described previously. The experiments were carried out with 10^{-4} and 10^{-3} molal dodecylammonium acetate solutions and the results were the same at both concentrations. In Table II the experimental data are tabulated according to whether or not the particular salt floats under these conditions.

Table II. Flotation of Alkali and Ammonium Halides with Dodecylammonium Acetate as Collector

	Li^+	Na^+	K^+	Rb^+	Cs^+	NH_4^+
F^-	No	No	Some			Yes
Cl^-	No	No	Yes	Yes	Yes	Yes
Br^-	No	No	Yes	Yes	Yes	Yes
I^-	No	Yes	Yes	Yes	Yes	Yes

Of the fluorides, there definitely was no flotation of LiF and NaF. Some air bubbles attached to KF particles, but it was difficult to tell about the floatability of KF because the solution appeared very syrupy. Flotation of NH_4F occurs under these conditions.

The experiments with the chlorides showed that LiCl and NaCl will not float with dodecylammonium acetate as collector, but KCl, RbCl, CsCl and NH_4Cl readily float.

Flotation tests with the bromides showed that KBr, RbBr, CsBr, and NH_4Br float readily with dodecylammonium acetate as collector and that LiBr does not float with this collector. However, although coarse NaBr particles do not float with dodecylammonium acetate as collector, very fine particles do float. No air bubbles appeared to attach themselves to the coarse NaBr particles.

Experiments with the iodides showed that NaI, KI, RbI, CsI and NH_4I all float with dodecylammonium acetate as collector but that LiI does not float. CsI appeared to respond to flotation a little more sluggishly than the other iodides.

Roughly, these data show that if the cationic collector can fit into the crystal lattice, the salt will float. To substantiate this, trimethyldodecylammonium chloride was used as the collector in a series of vacuum flotation tests with some of the chlorides. The radius of a tetramethylammonium ion is 3.06 A,[20] and the radius of the trimethyldodecylammonium ion is assumed to be the same as that of tetramethylammonium ions. The experimental results of tests run in 10^{-4} and 10^{-3} molal collector solutions are given in Table III.

Since ionic size apparently plays a role in the flotation of soluble halides with cationic collectors, it seemed reasonable to extend this to the flotation of alkaline-earth chlorides. Accordingly, vacuum flotation tests were run with 10^{-4} and 10^{-3} molal dodecylammonium acetate solutions with the alkaline-earth chlorides. Results are given in Table IV. Since Ba^{++} is slightly larger than K^+, and since the other alkaline-earth ions are considerably smaller, it follows that the flotation of alkaline-earth chlorides with dodecylammonium acetate depends upon ionic size also.

Table III. Flotation of Alkali and Ammonium Chlorides with Trimethyldodecylammonium Chloride as Collector

Salt	Flotation
LiCl	No
NaCl	No
KCl	No
NH_4Cl	No

Table IV. Flotation of Alkaline-Earth Chlorides with Dodecylammonium Acetate as Collector

Salt	Flotation
$MgCl_2$	No
$CaCl_2$	No
$SrCl_2$	No
$BaCl_2$	Yes

Contact Angle Studies on Halite with Dodecylammonium Acetate as Collector: The contact angle on a freshly cleaved halite crystal was measured in saturated NaCl brine containing various concentrations of dodecylammonium acetate as collector to acertain whether or not any adsorption of this collector does take place on halite. The experimental findings are tabulated in Table V. The contact angle of an air bubble on halite in a saturated NaCl brine containing dodecylammonium acetate as collector is zero at all concentrations of this collector.

Table V. Contact Angles on Halite with Dodecylammonium Acetate as Collector

Collector Addition in Mols per Liter of Brine	Contact Angle in Degrees
1×10^{-6}	0
5×10^{-6}	0
1×10^{-5}	0
5×10^{-5}	0
1×10^{-4}	0
5×10^{-4}	0
1×10^{-3}	0
5×10^{-3}	0

Contact Angle Studies on Sylvite with Dodecylammonium Acetate as Collector: To find out the magnitude of the contact angle on sylvite in a brine saturated with KCl and to see how it varies with the concentration of dodecylammonium chloride, a freshly cleaved crystal of Carlsbad sylvite was placed in a KCl brine containing different amounts of collector and the contact angle was measured. Five different angles were measured for each concentration and the average value of the contact angle is plotted against the logarithm of the collector addition in Fig. 1. At a collector concentration of only 10^{-6} mols per liter of brine, the contact

between an air bubble and a crystal of sylvite is only a slight cling. Above this concentration, the contact angle increases linearly with the logarithm of the collector addition until it reaches a maximum value of 55° at an addition of 5×10^{-3} mols of dodecylammonium acetate per liter of brine. Above this addition, the contact angle decreases sharply, possibly because of the formation of a second layer of collector ions being adsorbed in reverse orientation. The solutions began to become cloudy at additions of about 5×10^{-4} molar.

Contact Angle Measurements with Different Samples of Sylvite: To find out if any differences exist among various samples of sylvite, contact angles were measured with crystals of sylvite from Carlsbad, N. M., and Stassfurt, Germany, and chemically pure KCl in a saturated brine containing 10^{-4} and 10^{-3} molar dodecylammonium acetate. The experimental results are listed in Table VI. There is no difference within the experimental limits among different sylvite samples.

Reversibility of Adsorption: To study the reversibility of adsorption of dodecylammonium ions at the surface of sylvite, a crystal of sylvite was first placed in a brine containing 10^{-3} M dodecylammonium chloride. After the contact angle was measured the sylvite was taken from the more concentrated collector solution and was placed in a brine containing 10^{-4} M dodecylammonium chloride. In the 10^{-3} M solution, the contact angle was 47° and after the sylvite was placed in the 10^{-4} M collector solution, the contact angle was found to be 38°, which is the usual value found without preconditioning in the concentrated collector solution.

Table VI. Comparison Among Different Sylvite Samples

Salt	Collector Addition	Contact Angle in Degrees
Carlsbad sylvite	1×10^{-4}	37
	1×10^{-3}	46
Reagent KCl	1×10^{-4}	38
	1×10^{-3}	45
Stassfurt sylvite	1×10^{-4}	38
	1×10^{-3}	47

Effect of Chain-Length of the Collector Tail on the Contact Angle: A series of experiments was run using the primary amine acetates containing an even number of carbon atoms between 8 and 18 to

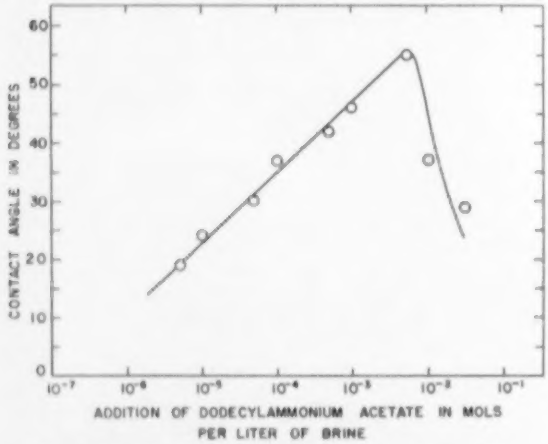


Fig. 1—Contact angle on sylvite as a function of the addition of dodecylammonium acetate in mols per liter of brine. (24 ± 2°C.)

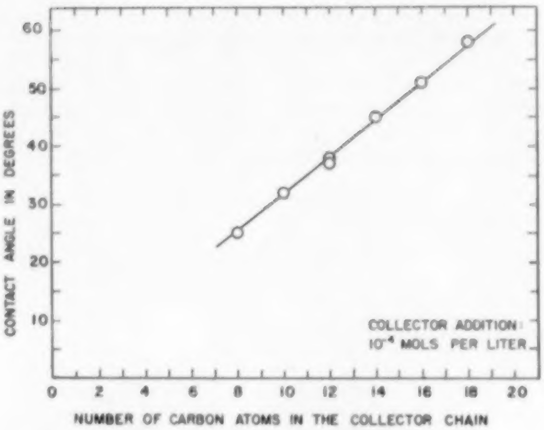


Fig. 2—Effect of the number of carbon atoms in the collector chain on the magnitude of the contact angle on sylvite. Flotation tests show that KCl floats with the 18, 16, 14, 12, 10, and 8-carbon amines but not with the 6, 4, and 2-carbon amines. (24 ± 2°C.)

find out how the number of carbon atoms in the chain of collector affects the contact angle on sylvite. For each experiment, the addition of collector was kept constant at 10^{-4} molar. Hexadecylammonium acetate and octadecylammonium acetate were difficult to dissolve. After the flasks had been shaken for three days, the collector appeared to be essentially dissolved. The experimental results are presented graphically in Fig. 2. This figure shows that the contact angle increases linearly with increased chain length.

Contact Angle Under Simulated Mill Conditions: Since in actual operating conditions, the solution is saturated not only with KCl, but also NaCl, one experiment was run with a brine saturated with both NaCl and KCl and a collector concentration of 10^{-4} mols dodecylammonium acetate per liter of brine. Under these conditions the contact angle on sylvite was found to be 38°, which is the same as that found in the absence of NaCl in the brine.

Effect of pH on the Contact Angle on Sylvite: To determine whether it is the aminium ion or the free amine which acts as the collector, the contact angle in brines containing 10^{-4} molar was measured as a function of pH. Hydrolysis of the amine takes place according to the following relation:[11]

$$RNH_2 + H_2O \rightleftharpoons RNH_3^+ + OH^-;\ K = 4.3 \times 10^{-4}$$

In dilute neutral or acidic solutions, the collector exists entirely as dodecylammonium ions but at pH 10 it is only 80 pct ionized and at pH 11 it is but 30 pct ionized. In the flotation of sylvite, a major effect of pH is to control the hydrolysis of the amine, whereas in the flotation of quartz, pH also controls the surface charge.[12, 13] Experimental data are presented graphically in Fig. 3. This figure shows that the contact angle decreases as the solutions become alkaline. At pH 11 the solution contains only 9×10^{-6} mols of dodecylammonium ions per liter and gives a contact angle of 28°. Fig. 1 shows that a brine containing 9×10^{-6} mols of dodecylammonium acetate per liter would give a contact angle of 23°.

Contact Angles of Sylvite with Trimethyldodecylammonium Chloride as Collector: Vacuum flotation tests showed that not only LiCl and NaCl do not float with trimethyldodecylammonium chloride as collector but also that KCl and NH_4Cl do not float

with this collector. To see if any of the large quaternary aminium ions are adsorbed at the sylvite surface, a series of contact angles were measured in brines saturated with KCl using trimethyldodecylammonium chloride as collector. In brines containing 10^{-4} and 10^{-3} molar collector, the contact angle on sylvite was found to be zero. Apparently this large cation cannot be adsorbed onto the sylvite surface.

Discussion of Results

Crystal Chemistry of the Alkali and Ammonium Halides: To understand how KCl and NaCl behave in the presence of a collector, it will be of interest to look at the crystal chemistry of the alkali and ammonium halides.

Because of the stability of the noble-gas configuration of electrons in the outer shell of an ion, alkali and halide ions retain their ionic structure even in solid crystals. A crystal of a substance made up of ions is held together by the electrostatic attraction between oppositely charged ions. Because of the spherical symmetry of the electron distribution of ions with noble-gas configurations, the interaction of an ion with other ions is independent of direction, and consequently the structure of the ionic crystal is largely determined by purely geometrical considerations.[14] Thus each ion is surrounded by the largest possible number of oppositely charged neighbors. However, for a given ion, the number of immediate neighbors is determined not only by the condition of geometric packing, but also by electroneutrality. For example, in sodium chloride the radius of the sodium ions is so much smaller than the radius of the chloride ions that there is room for a dozen or more sodium ions around each chloride ion.[14] However, electroneutrality requires that the number of sodium and chloride ions shall be equal and that the arrangement around each ion should be the same. Around each sodium ion, only six chloride ions, arranged regularly at the corners of an octahedron, can be accommodated. Hence this is the type of coordination not only around the sodium ions, but also around each chloride ion. Fig. 4a illustrates the sixfold coordination found in crystals of the sodium chloride structure.

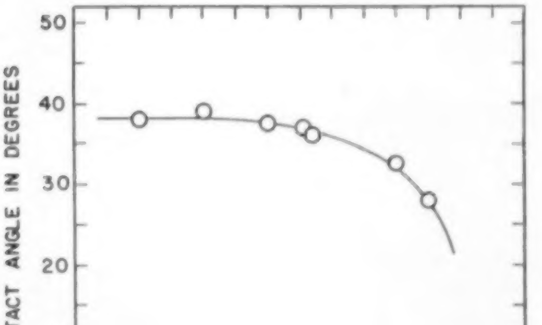
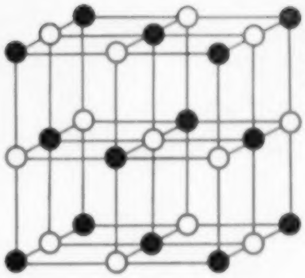


Fig. 3—Effect of pH on the contact angle on sylvite in brines containing 10^{-4} mols of dodecylammonium acetate per liter of brine. (24 ± 2°C.)



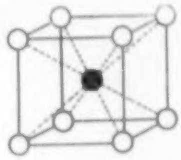


Fig. 4a (right)—The sodium chloride structures. Fig. 4b (below)—The cesium chloride crystal structures.

On the other hand, in cesium chloride the two ions are more nearly comparable in size, and eight chloride neighbors can be packed around the alkali metal ion. The structure is quite different from that illustrated in Fig. 4a in that one ion of each kind is now surrounded by eight neighbors symmetrically situated at the corners of a cube. Fig. 4b illustrates the eightfold coordination found in crystals with the cesium chloride structure. The interionic distances for crystals with the cesium chloride structure generally are about 3 pct greater than for those with the sodium chloride structure.[6]

These two different structures for substances so closely related chemically emphasize how far more important are geometrical than chemical considerations in determining crystal structure among some solids. From such crystal structures it can be seen that no molecule MX, as such, exists in the crystal, but that in sodium chloride, for example, each sodium ion is associated equally with six chloride neighbors.

The alkali halides all crystallize with the sodium chloride structure except CsCl, CsBr, and CsI, which have the cesium chloride structure. NH_4Cl, NH_4Br, and NH_4I crystallize with both the sodium chloride and cesium chloride structures, the former being stable above the transition temperatures (184.3°C, 137.8°C and −17.6°C respectively) and the latter below these temperatures.[6] NH_4F crystallizes with the wurtzite structure.[14]

In LiF each anion is approaching contact not only with the surrounding cations, but also with other anions. Consequently the repulsive forces are larger than they would be for either anion-cation or anion-anion contact alone, and the equilibrium with the attractive coulombic forces is reached with a lattice constant such that the cation-anion distance is larger than twice the anion radius. According to Pauling, this phenomenon of double repulsion is shown also by NaI, NaBr, and NaCl.[6]

In Fig. 5 the cube face layers (100) planes, of the lithium, sodium, potassium, and rubidium halides, are drawn with circles corresponding to the ionic radii but are drawn with the interionic distances observed by Pauling.[6] The (100) planes of CsF and NH_4I are also drawn. Fig. 5 includes diagrams of the (110) planes of CsCl, CsBr, CsI, NH_4Cl, and NH_4Br. The cesium salts are drawn with the observed interionic distances, whereas NH_4Cl and NH_4Br are drawn with the 3 pct increased interionic distance characteristic of the cesium chloride structure.

Role of Ionic Size in the Collection of Halides with Aminium Ions: Since ionic size appears to play such an important role in the crystal chemistry of the alkali halides, it seems that the mechanism of collection of these salts should be approached from the same veiwpoint. To illustrate schematically in

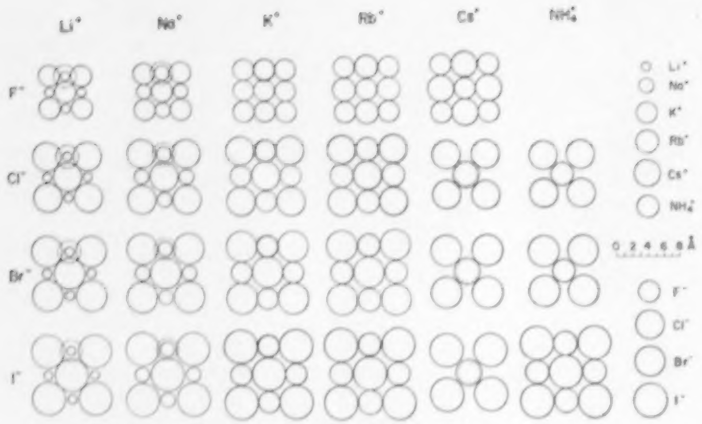


Fig. 5—Schematic representation of the (100) face of the alkali and ammonium halides with the sodium chloride structure and of the (110) face of those salts with the cesium chloride structure.

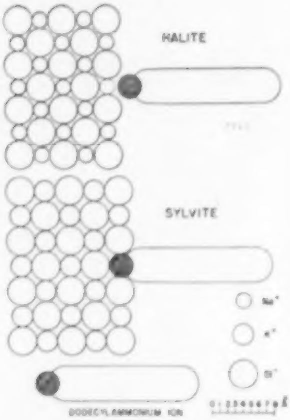


Fig. 6—Schematic representation of proposed mechanism of collection of sylvite and lack of collection of halite by dodecylammonium ions.

which crystals a surface cation can be replaced by a dodecylammonium ion, the ammonium ion is drawn with a dotted circle over one cation in each halide in Fig. 5. If ionic size controls the collection of the alkali and ammonium halides with aminium ions, observation of Fig. 5 should tell which salts will float. For example, it can be seen that all the halides of potassium, rubidium, cesium and ammonium should float because an aminium ion can fit into the position occupied by cations in these crystals. Furthermore, it can be seen that lithium and sodium ions are too small to be replaced except possibly in sodium iodide, in which crystal double repulsion spreads the ions apart. It should be remembered that monolayer coverage is unnecessary for flotation.[15] Flotation may take place with only a small percentage of the surface being coated with the collector.

The experimental results from vacuum flotation tests with the fluorides show that LiF and NaF do not float but that some flotation of KF and NH_4F does take place with primary aminium ions. Of the chlorides, LiCl and NaCl do not float but KCl, RbCl, CsCl, and NH_4Cl do float. Of the bromides, LiBr and NaBr do not float but KBr, RbBr, CsBr, and NH_4Br do float. However, these experiments showed that very fine NaBr particles do float. Pauling has stated that there is some double repulsion between ions in NaBr, and since very fine particles have a lot of corners and edges, this might add up to the fines being floatable. Of the iodides, NaI, KI, RbI, CsI, and NH_4I all float with dodecylammonium ions as collector. Sodium iodide floats because of the pronounced double repulsion in the crystal and because the iodide is a large squishy ion, all of which permits an aminium ion to squeeze into the lattice.

The radius of an ammonium ion is about 8 pct greater than the radius of a potassium ion. Apparently an ion only 8 pct larger than a potassium ion can fit into the KCl lattice. Van der Merwe showed that if the difference in atomic radii is less than 14 pct, it is possible for a monolayer of the second substance to be deposited at low temperature in exact fit on the substratum.[16] Furthermore, it is interesting to note that the Hume-Rothery rule states that the maximum difference in radius between ions in solid solution can be about 15 pct of the atomic radius of the solvent.[14] Since cesium salts down to potassium salts can be floated by primary aminium ions, this 15 pct must hold in the collection of halides with dodecylammonium ions.

Additional proof of the ionic size hypothesis lies in the flotation of the alkaline-earth chlorides. Only the barium ion is nearly the size of the ammonium, whereas the other alkaline-earth ions are much smaller. Vacuum flotation tests show that $BaCl_2$ floats and that $MgCl_2$, $CaCl_2$, and $SrCl_2$ do not float.

Since the charged head of a trimethyldodecylammonium is large, 3.06 A in radius, it would be expected that this reagent should float none of the alkali and ammonium halides, since it is too large to replace any surface cation. The experimental results show that quaternary aminium ions cannot collect any of the alkali chlorides, not even ammonium chloride. Further work could be done using monomethyl- and dimethyl-amine salts as collector.

The Collection of Sylvite with Primary Amine Salts: The surface of a soluble salt in a saturated solution is constantly changing at all times. At any given instant at the surface of a sylvite crystal, for example, potassium ions and chloride ions are leaving the surface for the solution and an equal number of ions are coming from the solution and depositing on the surface. Since each ion will be surrounded by the greatest possible number of oppositely charged ions if the crystal breaks with cubic cleavage, (100) faces would predominate. Since ions are constantly leaving the surface and being precipitated, collector ions will be incorporated into the lattice at the surface if they will fit.

The probable appearance of cube faces of halite and sylvite in a saturated brine containing some dodecylammonium chloride as collector is shown schematically in Fig. 6. All the ions are drawn to scale. It can be seen from this drawing that a dodecylammonium ion cannot replace a small sodium ion at the surface but that it can be incorporated into the sylvite lattice quite easily. Hence halite will not float, whereas sylvite does float with this collector. Actual incorporation of aminium ions into the lattice must be responsible for collection, because if adsorption held the ion next to the surface, as in the case of quartz, halite would float also. Furthermore, if only an adsorption mechanism con-

trolled the collection, trimethyldodecylammonium ions should act as collector.

Since the surface of sylvite is constantly dissolving and precipitating, it would be expected that the adsorption of collector ions at the surface should be reversible and dependent upon the concentration of collector ions in the bulk solution. Experiments have shown that the adsorption of collector ions is reversible, since sylvite conditioned first in 10^{-3} molar collector and placed in 10^{-4} molar collector afterwards gave the same contact angle as found in the less concentrated solution without prior conditioning in the concentrated solution. It was shown that the contact angle on sylvite increased with collector concentration up to a maximum angle of 55° at 5×10^{-3} M collector and fell off rapidly at higher collector concentrations, ostensibly because a second layer of dodecylammonium ions must have been held in reverse orientation through Van der Waals attraction among hydrocarbon chains. This would present a hydrophylic surface to the air bubble.

Apparently an aminium ion will collect a soluble halide if it can replace a cation in the lattice. Experiments have shown that the presence of large amounts of potassium, sodium, and hydrogen ions in solution do not affect the contact angle. Hence there must be little, if any, competition among these ions for the surface. If an aminium ion has deposited in the crystal lattice at the surface, it seems that association between the hydrocarbon tails may be the force holding these collector ions to the surface. Association of the hydrocarbon tails of collector ions has been shown to play an important part in the flotation of quartz with dodecylammonium ions.[16]

Summary and Conclusions

This experimental investigation has shown that ionic size controls the flotation of alkali and ammonium halides with the salts of primary amines as collector. If the collector ion can fit into the crystal lattice at the surface in place of a constituent cation, the particular halide can be floated.

Acknowledgments

For certain pieces of equipment and samples, the authors wish to thank J. P. Gries, A. E. McHugh, E. H. Stevens, and E. L. Tullis of the South Dakota School of Mines and Technology; H. J. Harwood of Armour & Co.; W. H. Dennen of the Massachusetts Institute of Technology; and F. H. Stewart of Southwest Potash Corp. The authors are grateful to A. M. Gaudin, P. L. de Bruyn, and F. F. Aplan of the Massachusetts Institute of Technology for their interest in this work and for reading the manuscript.

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Safety Factor Characteristic Curves For Mine Hoisting Ropes

by W. A. Boyer

MINE hoisting ropes can be loaded to capacity only when the strength of each component is exactly known. *Characteristic curves* provide this information. When load and rate of acceleration are specified for an individual rope, the characteristic curve assumes a predetermined shape.

The *factor of safety* is the value derived by dividing the breaking strength of the rope by the total stress on the rope where it goes over the sheave wheel. Total stress is made up of the weight of skip and cage, weight of ore, weight of rope between sheave and rope clevise or attachment, friction load, and acceleration force. Bending stresses are neglected in this discussion, as sheaves and drum diameters are assumed to be large enough to reduce these stresses to a negligible value.

Fig. 1 gives a family of characteristic curves for 1⅜-in. improved plow steel hoisting ropes of 6 x 19 or 6 x 21 construction with hemp core. The curves will hold for all rope sizes of this specified quality when they are subjected to an acceleration rate of 1.608 ft per sec². As acceleration rates of most deep shaft hoists are close to this value, this family of curves is presented as a standard.

Deviation is slight for other rope sizes. Note that for O, which is the most important, the deviation is within the width of the line on the graph. In a comparison of values for ropes of ⅝-in. and 2¼-in. diam the deviation ranges from a maximum 1.2 pct at 0 ft to 0.6 of 1 pct at 5000 ft. The value of 5.55 is the same for all ropes at 2500 ft.

The tabulated columns to the left of the curves are values of total connected load for each size of rope from ⅝-in. to 2¼-in. diam inclusive. With each of these connected loads the *safety factor characteristic curve* for the rope will be as indicated.

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Discussion of this paper, TP 4209A, may be sent (2 copies) to AIME before May 31, 1956. Manuscript, Aug. 17, 1955. Pacific Northwest Conference, Spokane, April 1955.

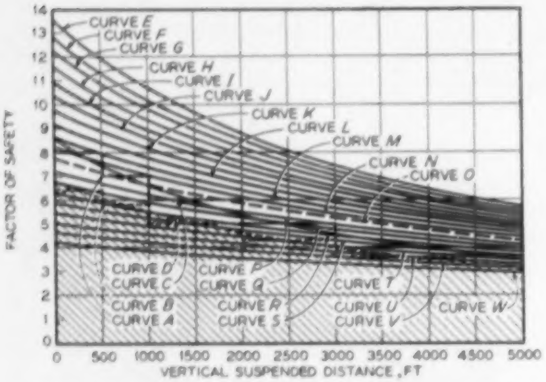

Fig. 1—Tabulated value under each size of rope listed to the left of the curves represents the connected load at end of rope which would make the safety factor value fall along the curve indicated. Values along the line *O-O* represent the maximum connected load for each size of new rope to fulfill the new *safety factor* requirements proposed here. Shaded area is that portion below curve *D* when rope must be discarded according to present factor of safety standards. The new proposed values of factor of safety at point of discard will follow approximately along curve *S*.

Improved plow steel ropes, 6x19 or 6x21 construction. Acceleration rate is 1.608 ft per sec², or 0.05 g.

Curve *O*—New safety factor values proposed by the author.

Curves *A* and *C*—Present safety factor values for new rope.

Curves *B* and *D*—Present safety factor values when rope should be discarded.

Note: Curves *A*, *B*, *C*, and *D* are more fully described in Fig. 1 of the author's article, "Safety Factor Characteristic Curves," *AIME Trans.*, October 1954, vol. 199, p. 990.

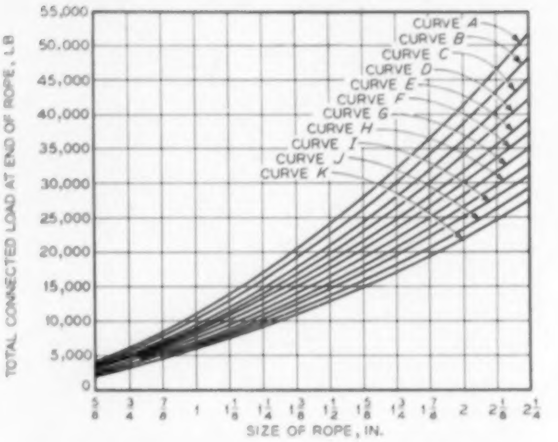

Fig. 2—Total suspended load at end of rope for the various sizes, with various rates of acceleration. Calculated to fulfill the factor of safety requirements proposed in the accompanying text. Improved plow steel rope, 6x19 or 6x21 construction.

Curve *A*—0.00 ft per sec², or g = 0.0.
Curve *B*—1.608 ft per sec², or g = 0.05.
Curve *C*—3.216 ft per sec², or g = 0.10.
Curve *D*—4.824 ft per sec², or g = 0.15.
Curve *E*—6.432 ft per sec², or g = 0.20.
Curve *F*—8.040 ft per sec², or g = 0.25.
Curve *G*—9.648 ft per sec², or g = 0.30.
Curve *H*—11.256 ft per sec², or g = 0.35
Curve *I*—12.846 ft per sec², or g = 0.40.
Curve *J*—14.472 ft per sec², or g = 0.45.
Curve *K*—16.080 ft per sec², or g = 0.50.

It is easy, therefore, to check on which curve a particular installation is operating. The line *O-O* indicates the allowable maximum connected load for the new rope to operate along the *safety factor* values as set up by the author. Adoption of the safety factor values along a characteristic curve eliminates calculations of rope factor of safety. With this method it is necessary only to compare connected load values with the allowable load for the size of rope under consideration.

When the family of curves were made up, the factors of safety were calculated for each connected load for various depths from 0 to 5000 ft at 500-ft intervals. These values were then plotted and the curve drawn. All calculations were started at the 2500-ft depth first and then worked both ways. This was done so that in a comparison, the curves for ropes of different sizes would coincide at the midpoint and deviation would be less at the extreme ends of the curves. The different curves of Fig. 1 were calculated by starting with various values of factor of safety at the 2500-ft depth and changing them in 0.25 increments. An exception is Curve *O*, which was taken at 5.55 so that it would coincide with the *new factor of safety standards* the author is proposing. The curves were calculated for various loadings for each of the different sizes of rope. When the family of curves was compared for the various sizes of ropes for the same grade of rope and rate of acceleration it was found that the curves would coincide very closely. This means that one family of characteristic curves will serve for all sizes of rope of the same quality operating under the same rate of acceleration.

Table I. Total Suspended Load, Lb, at End of Rope Correlated with Safety Factor Curves in Fig. 1

	5/8-In.	3/4-In.	7/8-In.	1-In.	1 1/8-In.	1 1/4-In.	1 3/8-In.	1 1/2-In.	1 5/8-In.	1 3/4-In.	1 7/8-In.	2-In.	2 1/8-In.	2 1/4-In.
E	2,401	3,417	4,592	5,952	7,449	9,129	10,925	12,905	14,901	17,274	19,496	22,096	24,544	27,369
F	2,529	3,599	4,839	6,273	7,853	9,625	11,522	13,611	15,723	18,226	20,579	23,324	25,919	28,905
G	2,662	3,794	5,103	6,616	8,284	10,154	12,158	14,365	16,600	19,242	21,735	24,635	27,385	30,544
H	2,813	4,003	5,385	6,982	8,744	10,720	12,839	15,171	17,537	20,328	22,969	26,036	28,953	32,295
I	2,969	4,226	5,687	7,374	9,236	11,326	13,568	16,034	18,540	21,491	24,292	27,537	30,633	34,172
J	3,137	4,466	6,011	7,796	9,768	11,977	14,351	16,961	19,619	22,741	25,713	29,150	32,436	36,187
K	3,319	4,724	6,361	8,249	10,339	12,678	15,194	17,960	20,780	24,087	27,244	30,960	34,379	38,358
L	3,515	5,003	6,738	8,739	10,955	13,434	16,106	19,038	22,035	25,540	28,896	32,762	36,477	40,703
M	3,727	5,306	7,147	9,270	11,623	14,255	17,092	20,206	23,393	27,116	30,687	34,794	38,750	43,242
N	3,957	5,634	7,592	9,847	12,349	15,148	18,164	21,476	24,870	28,826	32,633	37,002	41,221	46,003
O	*4,156*	*5,918*	*7,976*	*10,346*	*12,977*	*15,918*	*19,092*	*22,574*	*26,147*	*30,307*	*34,316*	*38,912*	*43,358*	*48,390*
P	4,484	6,385	8,608	11,166	14,009	17,186	20,615	24,379	28,246	32,739	37,081	42,050	46,868	52,312
Q	4,707	6,817	9,192	11,924	14,963	18,357	22,026	26,048	30,187	34,988	39,639	44,952	50,115	55,940
R	5,122	7,294	9,837	12,762	16,018	19,652	23,583	27,892	32,332	37,474	42,466	48,160	53,704	59,950
S	5,494	7,824	10,555	13,654	17,190	21,091	25,314	29,942	34,716	40,237	45,608	51,725	57,692	64,406
T	5,910	8,417	11,356	14,734	18,499	22,700	27,249	32,232	37,380	43,324	49,118	55,709	62,235	69,386
U	6,377	9,083	12,258	15,906	19,877	24,510	29,423	34,810	40,352	46,798	53,068	60,190	67,163	74,988
V	6,908	9,839	13,281	17,232	21,642	26,560	31,893	37,730	43,774	50,734	57,544	65,270	72,846	81,337
W	7,513	10,702	14,449	18,748	23,551	28,904	34,711	41,068	47,656	55,233	62,660	71,075	79,340	88,594

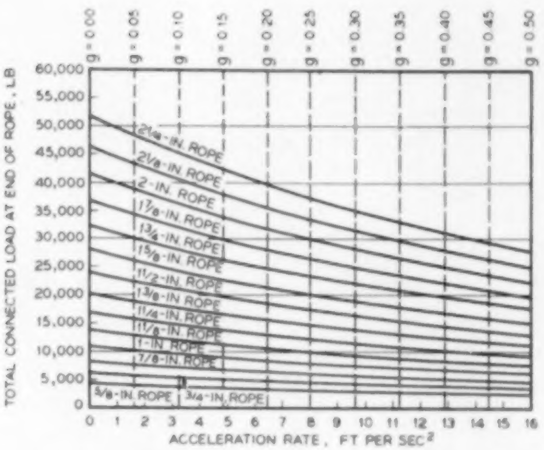

Fig. 3—Total suspended load at end of rope for the various sizes, with various rates of acceleration. Calculated to fulfill the factor of safety requirements as proposed by the author. Improved plow steel rope, 6x19 or 6x21 construction.

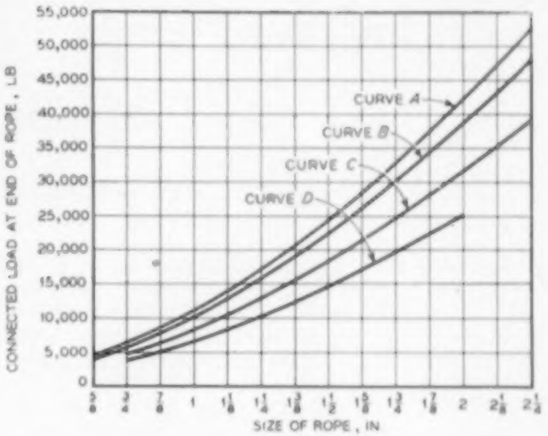

Fig. 4—Curves shown here offer direct comparison of values of connected load for various sizes and qualities of rope. The flattened strand rope is of the same quality of steel but of different construction.

Acceleration rate: 1.608 ft per sec², or 0.05 g.

Curve A—Flattened strand.
Curve B—Improved plow steel, 6x19 or 6x21.
Curve C—Plow steel, 6x19 or 6x21.
Curve D—Mild plow steel, 6x19 or 6x21.

The lightly sketched curve *A* represents the present recommended minimum factor of safety values for new ropes. Curve B represents the present recommended minimum factor of safety values when ropes must be discarded. *C* and *D* are smooth curves drawn through the midpoints of the steps of curves A and B respectively and more truly indicate the present required factor of safety for specific depths.

On comparison of curves *C* and *D* with the true characteristic curves for this quality of rope and with the permitted loading it is found that curves C and *D* are much steeper. For instance curve C follows along exactly for a much lower grade of rope or a 1¼-in. diam rope with a breaking strength of 75,000 lb. A present day 1¼-in. diam improved plow steel rope has a breaking strength of 129,200 lb.

The usual practice in figuring an installation is to determine first the size of load it is necessary to hoist in order to get out the required tonnage in the allotted time. From the value of the load the weight of skip and cage can be approximated. The size of rope is then determined, and even though the rope may not be fully loaded no further consideration is given to increase the size of load and skip in order to load the rope to capacity. With the use of the method proposed by the author the connected load value would be compared with the values along line *O-O* of Fig. 1 which would indicate instantly the necessary size of rope and also indicate the additional capacity that might be added to the skip to utilize the rope to the fullest extent and still be within the safety limits.

The safety factor characteristic curves for flattened strand improved plow steel quality, 6x30 type G or 6x27 type H ropes are almost identical with the curves shown in Fig. 1 if the connected load values along the line *O-O* are increased 9.04 pct.

In Figs. 2 and 3 the maximum loading is given for each of the different rope sizes, from ⅝-in. to 2¼-in. diam inclusive for improved plow steel 6x19, 6x21 type *U*, 6x25 type *W* construction, for all rates of acceleration from 0 to 16.08 ft per sec². On comparison with curve *A* of Fig. 2 or zero acceleration point of Fig. 3 the reduction of connected load due to inclusion of accelerating forces can be determined. When values were worked out for plotting the curves for each rate of acceleration, the loading was such that the factor of safety would be 5.55 at a depth of 2500 ft. This gave a loading for the rate of acceleration so that the rope would operate along a safety factor characteristic curve such as the author proposed for a standard.

Curves in Figs. 2 and 3 are plotted from the same values. In Fig. 2 the sizes of rope are used as abscissa and in Fig. 3 the rates of acceleration are used as abscissa. The value of the maximum connected load for each size of rope for any rate of acceleration is probably found most readily from Fig. 3.

All the tabulated values and curves have been calculated on the assumption that the ropes are to be working in vertical shafts, or in other words, at a 90° angle from the surface horizontal. All the values can also be used for slopes or shafts at an angle less than 90° with respect to the surface horizontal. In that case the value would be divided by the sine of the angle included by the surface horizontal and the centerline of the shaft. The angle of slope would be increased slightly from the actual value so as to include a coefficient of friction component which would be more apparent in slope or incline installations.

The author would like to point out that the connected load values given in Table I or obtained from the curves of Figs. 2 and 3 would be the same irrespective of depth of shaft. Variation in rope factor of safety with increase in depth is automatically compensated by the inherent characteristic curve of the rope.

In Fig. 4 a set of curves offers comparison of the allowable connected load for hoisting ropes of various qualities or types of construction. These curves show the advancement that has been made in rope load carrying capacity. Curve *C* is for a rope of higher steel quality than the rope of curve *D*. Curve *B* is for a rope with a higher grade of steel than that of curve C. Curve *A* is for a rope of the same quality of steel as the rope in curve *B* but with a more recent type of construction.

Improved Contact Angle Apparatus for Flotation Research

by Donald W. McGlashan and Kenneth N. McLeod

IN the use of free bubbles with precise temperature control and continuous pH measurement, the contact angle apparatus differs from all previous equipment. Experimental procedures differ sharply from the captive bubble method[1, 2] of introducing the gaseous phase in the three-phase system. When freed from the tube and captured by a solid surface, a bubble more closely conforms to the spherical ideal set up by the usual mathematical analysis of surface free energy or surface tensional relationships of the forces involved.

The assembled equipment, Fig. 1, departs from the usual contact angle apparatus in providing: 1) water circulation for fixing temperature, 2) instruments to measure and record pH during testing, and 3) a system for producing and using free bubbles. The apparatus allows testing over a broad range of closely controlled conditions, and the use of free bubbles gives contact angles independent of operator manipulation, which was unavoidable in the older captive bubble method. A bubble released below the specimen rises through the liquid and contacts the mineral. If the mineral is gas-avid the bubble adheres, developing a finite contact angle.

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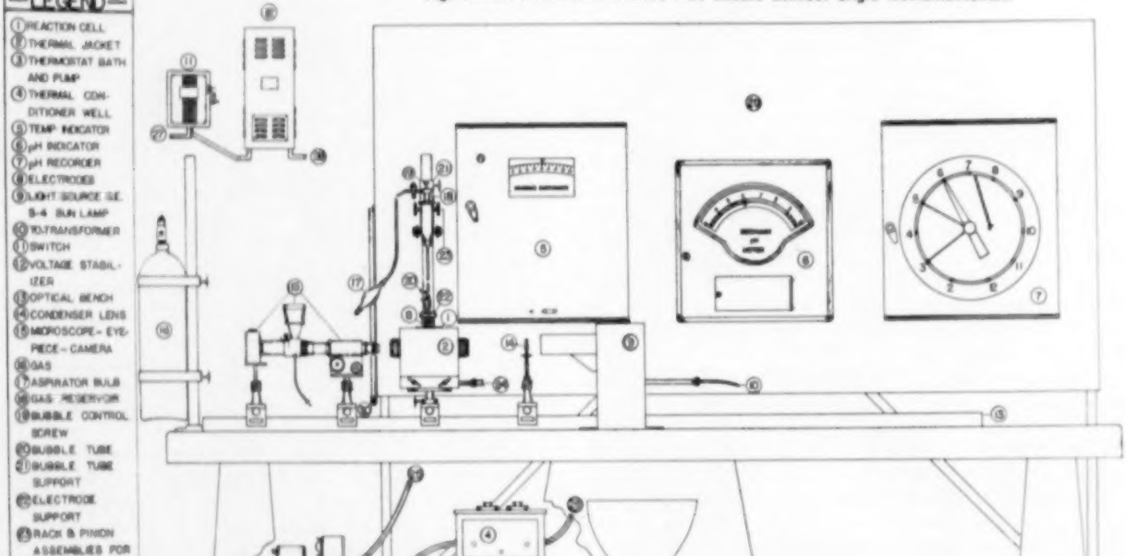
Discussion of this paper, TP 4211B, may be sent (2 copies) to AIME before May 31, 1956. Manuscript, March 14, 1955. New York Meeting, February 1956.

Reaction cells (1), 2x3⅜x5 3/16 in. ID, are of pyrex glass 5 mm thick. To prevent optical distortion, the sides are optical flats and are planed parallel to within five wave lengths of light over the entire surface. Special cement is required to withstand chemical attack and thermal shock. The cell is placed in a thermal jacket (2) made of welded aluminum. A water seal for the optical ports is made by using O-rings tightened against the flats of the cell by fine-threaded and adjustable ports, Fig. 2. In-flow water is discharged downward from a manifold located near the bottom of the cell jacket. Water overflows the weirs at each end of the cell jacket, discharging from both sides to a collecting manifold (24). A hose (26) leads the overflow water to the thermal-conditioner well (4), which is recessed to accommodate a reaction cell and two beakers (150 ml). Both the thermal jacket and the thermal-conditioner well are insulated to minimize the effects of temperature differentials. Water, discharging from the thermal-conditioner well, returns to a thermostat bath and pump (3), from which it is pumped to the thermal jacket. The thermostat bath permits temperatures to be adjusted from 1° or 2°C to 80°C and will maintain the selected temperature within 0.2°C.

An electrode support (22) groups the four electrodes (8) so that they fit conveniently into the reaction cell, Fig. 2. This assembly holds the glass electrode, the calomel electrode, the temperature-

Fig. 1—Environment-controlled free bubble contact angle instrumentation.



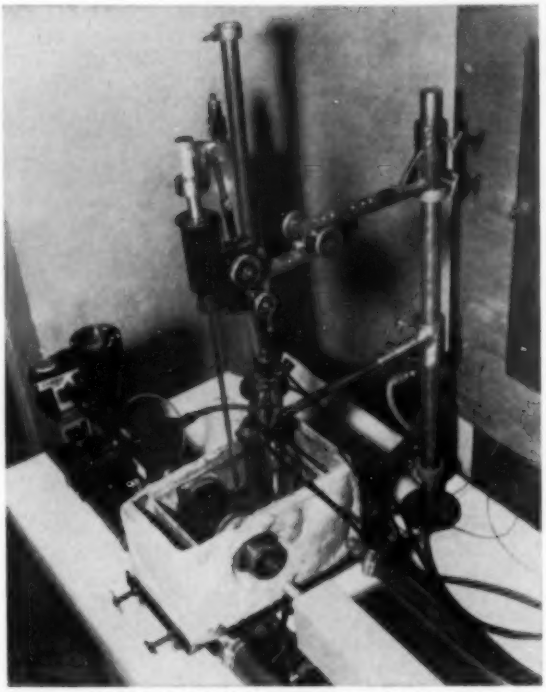


Fig. 2—Free bubble contact angle assembly.

compensating thermocouple for the pH system, and the thermocouple for temperature indication. The temperature-indicating thermocouple is glass-insulated 24-gage iron-constantan which is cemented in pyrex glass tubing. The exposed spot-welded junction of the thermocouple is given a light plastic coating to prevent corrosion of the wire. The thermocouple is connected to the electronic temperature indicator (5),* which has a temperature range from

* Brown Electronick Precision Indicating Single Point Potentiometer.

0° to 100°C and 0.2°C scale subdivisions. Leads from the three electrodes for the pH system go to a Beckman Model R indicating amplifier (6), which has a range from pH 3 to 10. The amplifier is connected to a Brown Electronick Recording Potentiometer (7) with a 12-in. diam chart and 24-hr period. The recording potentiometer extends the pH range, giving a working range from pH 2 to 12 in 0.1 subdivisions. Adjustment and placement of the electrodes are provided by a sleeve-clamp and vertical support rod, Fig. 2.

An optical bench is used to support the light source (9), the condensing lens (14), the thermal jacket, and the microscope, eyepiece-viewer, and 35-mm camera (15). Suitable mounts on standard carriages have been designed to hold each piece of apparatus, permitting easy optical alignment. There is no operational time limit on illumination, which is provided by a high intensity mercury lamp with a steady arc requiring no adjustment. Satisfactory pictures are obtained with exposures ranging from 1/10 to 1/100 sec. The microscope is equipped with a 48-mm 2x objective and 10x ocular. The ocular is fitted with either a reticulocyte disk or a micrometer disk. The reticulocyte disk has two parallel lines 2 mm apart. Focusing of the microscope is facilitated by the eyepiece-viewer, which has an engraved outline defining frame-size for 35-mm film.

Fig. 3 illustrates the bubble tube assembly. Particularly important is the gas reservoir, which holds enough gas to produce 20 bubbles or more. The reservoir is charged with gas from a cylinder (16) through a needle valve or with air by an aspirator bulb (17). In either case closing of the stopcock completes the charging operation. Positioning of the bubble tube in the reaction cell below the inverted specimen is carried out by adjustment of two sets of rack-and-pinions, one providing horizontal alignment, the other insuring vertical placement with respect to the inverted specimen and to the light path. After positioning, tightening the control produces a bubble on the tip of the bubble tube. While the bubble is held on the tip, the control screw permits regulation of the bubble size. Capillary glass tubing with a bore of approximately 0.5 mm is used for the bubble tube, which is bent at the tip as shown in Fig. 3. To prepare the tip, the capillary tubing is first cut and then ground and polished to eliminate rough edges and provide for easy release of the bubble.

Minerals for study are first roughly shaped and then briquetted in plastic (DuPont Composition HG-4F). Enough plastic is used so that the upper end of the briquette can be drilled and tapped to accommodate the stem of the specimen holder. Both machine-polished and hand-polished specimens are used, depending on the physical characteristics of the specimen. The mineral specimen is suspended in the test solution by a holder rested on the edges of the reaction cell. Final adjustment is easily made by turning the stem and sliding the specimen holder to the desired position with respect to the optical ports, light path, and engraved picture frame.

Experimental Procedure: A solution containing a surface active agent is placed in the reaction cell, which has a volume slightly in excess of 500 ml. The reaction cell is then transferred to the thermal jacket, O-rings are inserted into the recessed grooves of

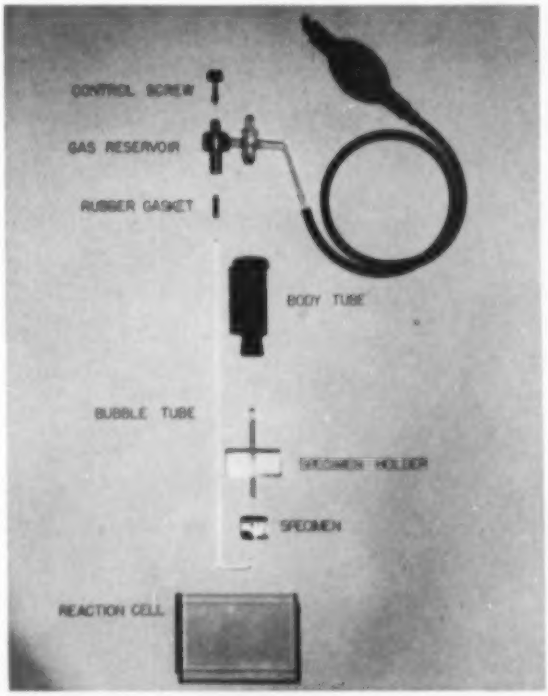


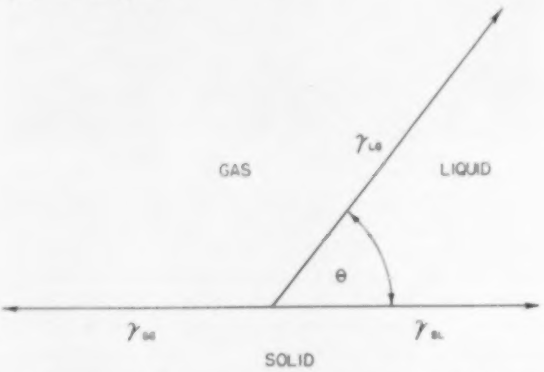
Fig. 3—Bubble tube assembly, specimen holder, and reaction cell.

the optical ports, and the ports are tightened against the cell walls. After standardizing with an appropriate pH buffer, the rinsed electrodes are positioned in the reaction cell. Time required for the circulating system to bring the test solution to temperature depends on a predetermined setting for this environmental factor. Once the system comes to the desired temperature, it may be maintained within ±0.2°C. In the low temperature range, fogging of the optical port is prevented by directing a stream of air against the cell wall at the porthole nearest the microscope. At the predetermined temperature, pH adjustments are made to bring this factor to the desired setting. Continuous pH indicating and recording enables very close control of this factor. A Cenco-duNouy tensiometer is used to measure the surface tension of the aqueous solution. Capillary methods of measuring surface tension of the solution in the reaction cell have been used and give greater accuracy than the tensiometer.

A stream of distilled water and a cloth-covered glass lap are used to buff the previously polished specimen on a variable speed polishing wheel. Experimentation will determine the type of polishing cloth and the abrasive to be used on various minerals. After buffing, the mineral specimen is thoroughly washed and immersed in distilled or demineralized water. It is then tested for *cleanliness* or water wettability. For this critically important test, the mineral specimen is placed in a reaction cell containing distilled or demineralized water and a bubble is either released or pressed on the mineral surface. A pressed bubble may be used to survey the entire surface of the mineral by moving the specimen holder. If the surface is contaminated the bubble will cling, indicating that buffing must be repeated. While the contact angle apparatus described herein can be used for *cleanliness* testing, an auxiliary apparatus is convenient and speeds the preparatory steps. On completion of this test, the mineral specimen is returned to the storage vessel (150 ml beaker) and placed in the thermal-conditioner well for temperature equalizing.

After the mineral specimen has been temperature-conditioned, it is transferred to the reaction cell and positioned with respect to the light path and the engraved outline of the film frame-size on the eyepiece-viewer. This places the mineral surface approximately 3 cm below the liquid surface. The tip of the bubble tube is moved directly under the mineral. A bubble is produced and sized. Tapping the tube gently releases the bubble. If the surface active agent has altered the surface of the mineral, making it air-avid, the free bubble will be captured and a finite contact angle formed. The camera then records the captured bubble for later measurement of the contact angle. On the other hand, a bubble will roll across an inactive mineral surface, either being captured by the plastic mounting or escaping to the surface of the solution.

Bubbles captured under equilibrium conditions always have a finite contact angle. The magnitude of the angle is determined from the photographic negative by measurement of the chord and the diameter of the segmented spherical bubble. The contact angle, measured through the liquid phase, is equal to one half the angle subtended by the chord. This relation, Fig. 4, is capable of simple geometric proof:



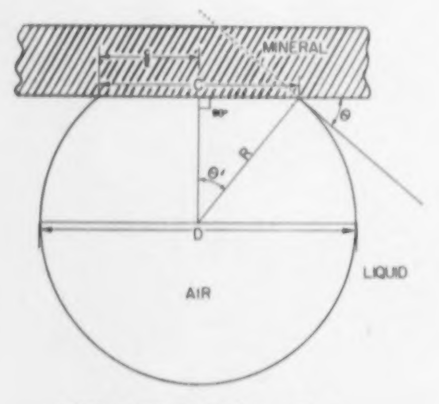


Fig. 4—Chord diameter and angle relation.

The sine of the contact angle is equal to the ratio of the chord to the diameter. When the sine has been calculated, a table of natural functions gives the angular value. A 7x focusing magnifier fitted with a 20 mm scale with 0.1 subdivisions is used for measuring. For convenience, the negative is placed on an illuminated ground glass screen.

Theoretical Considerations: Contact angle is a definite and real physical quantity characteristic of a three-phase system. Conventionally, it is measured within the liquid, and it is the angle between the liquid-solid surface and the liquid-gas surface. Dupre[3] first formulated the relationships between the surface energies or surface tensions to obtain the adhesional work involved in the approach of two unlike surfaces. The work of adhesion (W_a) makes it possible to express the energy difference in the three-phase system as Eq. 1:

$$W_a = \gamma_{lg}\,(1 + \cos\theta). \quad [1]$$

The decrease in surface free energy (ΔE_s) of the interfacial system has more thermodynamic significance than work of adhesion. ΔE_s is also a function of θ, the contact angle and γ_{lg}, the surface tension of the liquid-gas interface. When surfaces of unit area are compared:[4-6]

$$\Delta E_s = \gamma_{lg}\,(1 - \cos\theta). \quad [2]$$

When the solid surface is completely liquid-wet, the work of adhesion (Eq. 1) between the liquid and the solid is a maximum, $W_a = 2\,\gamma_{lg}$. The contact angle is zero and the gas will not be attracted

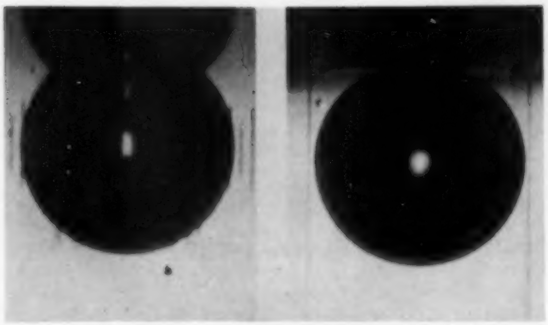


Fig. 5a (left)—Free bubbles in stable equilibrium on mineral surfaces and Fig. 5b (right)—in metastable equilibrium on mineral surfaces.

to the solid surface. Under this condition, there is no change in surface energy, $\Delta E_s = 0$ (Eq. 2).

Alteration of the surface environment can change the interfacial relationships. The liquid no longer attracts the solid as much as it attracts itself. In other words, the gas may displace the liquid from the solid surface. This displacement is manifested by the establishment of a finite contact angle. In mathematical terms, the work of adhesion is decreased, $W_a < 2\,\gamma_{lg}$. Also, the formation of a gas-solid interface indicates that the surface free energy has decreased. Since tensional forces of the solid-gas (γ_{sg}) and solid-liquid (γ_{sl}) cannot be measured, contact angle provides a means of following changes in surface energy occurring in the three-phase system. Fig. 5a shows an air bubble in equilibrium with the mineral and water phases. In this example, the magnitude of the contact angle indicates appreciable decrease in surface free energy. Fig. 5b shows another bubble in metastable equilibrium. The very small contact angle demonstrates negligible change in surface energy. Because of the buoyancy effect and the possibility of optical aberration, it is doubtful that a finite contact angle has been developed.

In the techniques developed for free bubbles, the situation illustrated by Fig. 5b is termed an *apparent contact*. A bubble in this condition will escape very readily, indicating lack of air-avidity on the mineral surface. On the other hand, bubbles captured by a definitely active surface are not easily dislodged. On a properly conditioned mineral surface, an equilibrium contact angle is established almost instantaneously. Bubbles placed on a mineral surface prior to its reaching equilibrium with the environment will give contact angles increasing in magnitude until environmental equilibrium has been achieved. The time factor for attainment of equilibrium between solid and liquid depends on the composition of the phases and on the temperature, but for any given set of conditions this time factor is constant. It must be determined before measurements under equilibrium conditions can be made. These considerations do not imply that all bubbles measured by the free bubble technique will give angles of contact that are precisely the same. Magnitude of the observed deviation is usually less than ±5°. Several factors are responsible for the deviation between different bubbles. Normal experimental errors undoubtedly account for some of the differences in angular values. Reducing these to a minimum will not eliminate the differences, since the bubble is *free* to seek a parking space on the mineral surface. The heterogeneous nature of a mineral surface with respect to composition and activity means that the bubbles are likely to be captured by areas having slightly different mean activities.

Table I. Data for Point A, Fig. 6

Picture No.	Exposure Time, Min	pH	°C	Contact Angle
1	5	10.00	17.0	21° 31'
2	10	10.00	17.0	19° 31'
3	15	10.00	17.0	15° 52'
4	22	10.00	17.1	18° 6'
5	25	10.02	17.2	18° 2'
6	31	10.02	17.2	19° 1'
7	38	10.00	17.0	18° 24'
8	40	10.00	17.0	23° 25'
9	45	10.00	17.1	22° 24'
Average		10.00	17.1	19° 33'

To minimize errors caused by bubbles of non-uniform size, the guide lines of the reticulocyte disk provide a convenient method for sizing bubbles. While Bashforth and Adams[7] have tables giving the necessary information for calculating the contact angle from the dimensions of a drop, Mack[8] and Mack and Lee[9] pointed out that the units tabulated by Bashforth and Adams cannot be readily calculated from experimental data. Wark[10] in review of Bashforth and Adams states: "For bubbles of air in water it is only over a limited range that the calcu-

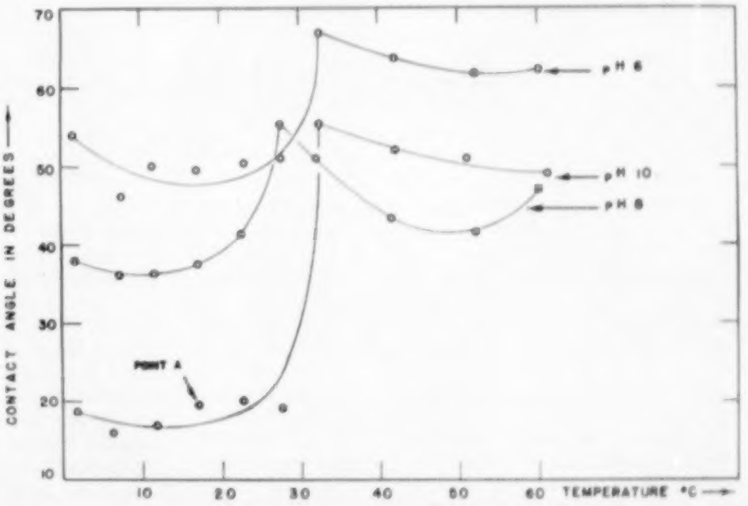


Fig. 6—Contact angle-temperature curves for activated sphalerite with potassium ethyl xanthate.

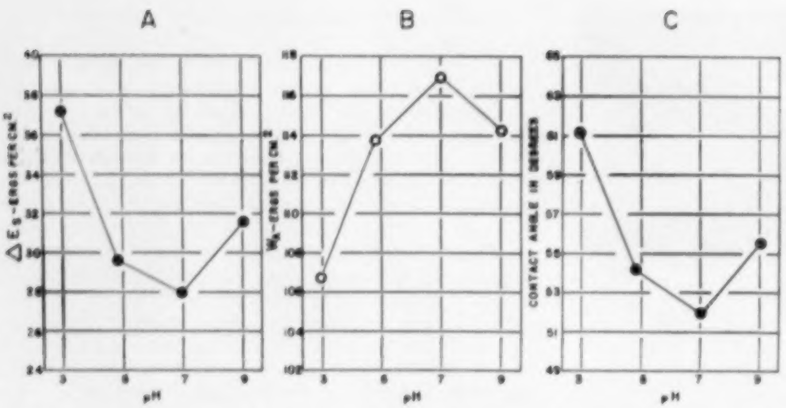


Fig. 7 — Relationship between A—surface energy, B—work of adhesion, and C—contact angle for chalcopyrite exposed to surface active agent.[12]

lations are possible." Through experimentation, an optimum bubble size was obtained that was large enough for accurate measurement and small enough to minimize distortion. For bubbles approaching sphericity, the contact angle is conveniently determined from a photographic negative by measurement of chord and diameter. Contact angle measurement by a position angle micrometer is more accurate, but the difference between methods is usually less than other experimental errors.

Experimental Data: Fig. 6 presents data graphically to illustrate effects of temperature on contact angle for the system activated sphalerite-potassium ethyl xanthate. Table I contains data for point A of Fig. 6.

Fig. 7 indicates three ways of presenting the data. It also shows relationships between surface free energy, work of adhesion, and contact angle.

Acknowledgments

Procurement of instruments and development of apparatus were made possible by a grant from the Research Corp. supplemented by funds from the Montana School of Mines. The authors express their appreciation to W. A. Woods, Western Field Office, Research Corp.; J. R. Van Pelt, president, Montana School of Mines; and Rayworth F. Howe, manager, Christie Transfer & Storage Co., for their invaluable encouragement and assistance. Acknowledgment and thanks are also extended to the following former students for assistance and suggestions: Robert E. Baarson, Chemical Div., Armour & Co.; Frank F. Aplan, graduate fellow, Massachusetts Institute of Technology; Wilbur J. Guay, Howe Sound Co., and Walter G. Parker, Western Machinery Co. Suggestions from Robert R. Beebe, research assistant, and Darrell E. Bodie and Emil A. Kovacevich, research fellows, were most helpful.

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Truth and Fallacy About a Serious Problem

Acid Coal Mine Drainage

by S. A. Braley

DRAINAGE of acid mine water into surface streams of coal mining areas is one of the most serious problems of stream pollution, since there is no known method that completely prevents its forming and no economically feasible treatment after it has formed. The mine acid problem differs from other pollution hazards because acid production does not end with cessation of mining but actually becomes more evident when pumping is stopped.

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Discussion of this paper, TP 4204F, may be sent (2 copies) to AIME before May 31, 1956. Manuscript, Feb. 15, 1955. Chicago Meeting, February 1956.

The acid problem in the Pennsylvania, Ohio, West Virginia, and Kentucky bituminous coal areas of the U. S. began with the first operation in 1761, and the volume of acid discharge today is largely the cumulative result of years of mining. Provided they lie above the drainage area, the oldest mines, now long abandoned, are almost without exception producing acid in the same manner as those now operating or recently abandoned.

It is well to consider some of the many procedures that have been used to determine the composition

or deleterious effect of acid mine drainage. This requires knowledge of its chemical composition. Generally speaking, it is a water solution of ferrous and ferric iron, aluminum, calcium, and magnesium sulfates. There may also be manganese, sodium, and potassium, and occasionally other metallic elements as chlorides, carbonates, or sulfates.

Major constituents fall into two specific classes: those that react with water to produce an acid solution, namely, the sulfates of iron and aluminum, and those that are merely dissolved in the water, or the calcium and magnesium sulfates.

The effective acid is the result of an original oxidation of sulfuritic materials associated with the coal measures, usually FeS_2 or pyrite. This reaction produces ferrous sulfate and sulfuric acid. Subsequent reaction of the sulfuric acid with the rock, shales, and limestones also associated with the coal measures converts it to aluminum, calcium, and magnesium sulfates. Analyses of hundreds of samples of acid mine discharge show that the amount of sulfate present is chemically equivalent to the sum of the metallic elements. It is usually considered, therefore, that there is no *free acid*.

Most frequently used to determine the apparent acidity of such a solution, the pH is only a quantitative value and does not indicate quantity of acid per volume. Again, the term *free acid* is only a measure of the amount of reaction that has taken place between the iron and aluminum salts and the water in which they are dissolved and is a value that changes with the state of oxidation of the iron, the temperature of the solution, and the degree of dilution.

One analysis, which can be duplicated for any sample, is important in determining not only the immediate effect of acid water entering the stream but also the ultimate effect. *Total acidity* is determined by titration in hot solution to a neutral end point, or pH 7.0, or for routine analysis and control, to phenolphthalein indicator.

Over the past 200 years since coal has been mined in the U. S., many attempts have been made to solve the problem of acid-forming reaction. These include neutralizing with alkalies, usually lime or limestone, sealing* to prevent entrance of air necessary for oxidation, and lagooning the discharge. Although neutralizing may be used as an emergency measure and sealing as a safety measure, none of these methods have been successful, and most have been costly. Even if some effective treatment were known, the preferred procedure would be to prevent formation of acid at the source.

In recent months the press has published reports that formation of acid can be stopped by use of inhibitors. This proposal is not new, but no such inhibitor has come to the attention of the writer or his colleagues. Moreover, application at the acid-producing spots in abandoned mines, the worst offenders, presents almost insurmountable difficulties, as such mines, being without maintenance or ventilation, are inaccessible because of roof-falls or gas. There are three possible methods: 1) introduction into the water entering the mine, 2) application on the surface of the overburden with the possibility that the inhibitor will be carried into the mine by water seeping through the strata, and 3) introduction as a gas that would permeate the mine by convection.

The first of these, introduction of an inhibitor into the water entering a mine, is not feasible because it is seldom possible to determine the point of entry or to trace the flow. Water seeps through walls or floor rather than through the roof, and flowing water usually contacts only very small areas of the acid-producing surface.

Application to the surface above the mine presents a number of problems. The inhibitor must be absorbed by the soil, and not allowed to run off to contaminate streams. The overburden must be porous enough to permit direct vertical penetration to the worked area underneath. The inhibitor must not react with the overburden or be adsorbed by it, but must reach the workings in the desired concentration. Finally, it must have no deleterious effect on the surface vegetation. Because of the geological strata in the mine cover, it is very questionable if a specific point of application to the surface would under any condition eventually seep into the mine excavation at the point to be treated.

The difficulties involved in introducing an inhibitor as a gas can be illustrated by an experiment conducted in 1950. Neutralization of acid and deposition of its iron content at the point of formation is evidenced by the brown areas and streaks on roof and walls in rock-dusted sections of operating mines. These brown areas appear so heavily coated that permeation of air to the oxidizable sulfuritic material is substantially retarded. If a gaseous alkali could be made to permeate all exposed sections of an abandoned mine the amount of acid produced would be materially decreased. An experiment of this sort was devised in which liquid ammonia, which vaporizes to gaseous ammonia at a temperature of −33° to −35°C, was introduced as the alkali or neutralizing agent.

A small abandoned mine that had been under observation for two years was selected for the purpose. During the period of observation, flow and acidity were measured at monthly intervals. This mine had a maximum cover of about 200 ft. Very few falls were apparent in the accessible areas, accessibility being determined by the oxygen content of the atmosphere. Maximum human penetration was about 700 ft. According to underground maps the entire mine was open as far down as a borehole about 1700 ft from the entry. Oxygen content of the mine atmosphere at the borehole varied from 2 or 3 pct to 21 pct, indicating that the borehole opened into the mine and that the atmosphere was as variable as at other accessible points.

A ¼-in. iron pipe placed in the borehole met an obstruction at a depth of 205 ft. It was assumed that this obstruction was the bottom of the mine excavation, since the mine map indicated a borehole depth of 200 ft. Through this pipe 500 lb of liquid ammonia were introduced, equal to about 10,500 cu ft of ammonia gas. This amount was sufficient to neutralize all the acid delivered by the mine for one week. Following introduction of ammonia hourly samples of the discharge were taken for 24 hr, daily samples for three weeks, and weekly samples thereafter. No decrease in acid delivery was observed at any time, and no odor of ammonia at any accessible point in the mine.

This experiment indicated the futility of assuming that gaseous materials can penetrate thoroughly an unventilated abandoned mine or that all the drainage from an abandoned mine exits at one point. Absence of uniform convection is also indi-

* Some deep shaft mines seal themselves by filling with water, completely submerging the acid-producing areas.

cated by the fact that in an operating mine abandoned areas adjacent to regular air courses are frequently low in oxygen. This condition is probably the result of closure of passageways by falls, which may or may not be permeable by water or vapor, or closure by water seals at the falls. Since there is no forced ventilation the gas must travel by convection, and under these conditions efficiency of the barriers increases.

Table I. Effect of Phosphates, Chromate, and Alkalies on Reaction of Sulfuritic Material with Oxygen

Solution	Days of Submersion at pH				
	0	5	14	28	42
N/100 $K_2Cr_2O_7$	4.69	5.20	3.30	2.6	2.18
N/100 K_2CrO_4	7.57	8.52	7.10	3.85	2.80
N/100 $Cu_2Cl_2(NH_3)_x$	11.22	9.72	8.93	7.7	4.22
H_2O	5.69	3.10	2.70	2.2	1.85

Table II. Extent of Oxidation Indicated by Sulfate Produced

Solution	SO_3, Ppm	
	28 Days	42 Days
N/100 $K_2Cr_2O_7$	840	800
N/100 K_2CrO_4	816	1184
N/100 $Cu_2Cl_2(NH_3)_x$	1176	1700
H_2O	560	864

Table III. Rate of Sulfate Production Correlated with Change in pH

Solution	Initial pH	Decrease in pH	SO_3(Ppm) 42 Days
N/100 $K_2Cr_2O_7$	4.69	2.51	800
H_2O	5.69	3.84	864
N/100 K_2CrO_4	7.54	4.77	1184
N/100 $Cu_2Cl_2(NH_3)_x$	11.22	7.00	1700

The volume of ammonia gas introduced into the mine was such that it should have been apparent at the entry had there been no obstruction between the borehole and the entry. Several efforts were made to reach the foot of the borehole from the entry, but a point was always reached where the atmosphere became so deficient in oxygen that safety lamps were extinguished. It is possible that an amount of acid equivalent to the amount of ammonia introduced had been neutralized in some area of the mine, but that area was separated from the accessible area by falls and thus there was no permeation of the ammonia or the water. This condition has also been found in mines assumed to be filled with water, but drillholes into some areas have shown them to be perfectly dry, probably because the point of entrance of the drillhole was sealed from the flooded portion by falls.

While this experiment demonstrated the accepted concept of conditions in abandoned mines, it seemed advisable to explore further the use of inhibitors to oxidate pyritic material. Use of phosphates to protect steel surfaces against corrosion is well known. An iron phosphate coating is formed on the clean steel surface which is semi-impervious to oxygen of the air. The action of the phosphate, however, is a chemical reaction with the steel. Nitric acid, which apparently produces an oxide film impervious to air, is used to passivate bright finish stainless steel surfaces. The rate of corrosion of zinc-coated steels is materially decreased by treatment of the surface with a chromate compound. In general, the rate of oxidation of metal surfaces is decreased by addition of alkaline materials with an increase in pH.

Following review of these reactions an experiment was planned to determine the effect of various phosphates, chromates, and alkalies on the reaction of sulfuritic material with oxygen in the air. Fifty grams of sulfuritic material were submerged in 200 ml of water and N/100 solutions of potassium chromate, potassium dichromate, and ammonical cuprous chloride. The pH of the solution was determined at the time the sulfuritic material was submerged. The pH and sulfate produced in the solution were determined at subsequent intervals. Data for pH is shown in Table I.

The sulfate produced was used as a criterion of the extent of the oxidation. These are reported in Table II as SO_3 in ppm.

Rate of sulfate production can be correlated with the change in pH in that the highest sulfate production is obtained in those solutions showing the greatest initial decrease in pH, as illustrated in Table III.

Table IV. Tests on 50 G of Sulfuritic Material

Solution	G per Liter of Salt in Solution
N/100 Na_3PO_4	1.2673 $Na_3PO_4 12 H_2O$
N/100 Na_2PO_4	0.9536 $Na_2HPO_4 7H_2O$
N/100 NaH_2PO_4	0.4600 NaH_2PO_4 H_2O
N/100 K_2CrO_4	0.6473 K_2CrO_4
H_2O	Distilled
N/100 $K_2Cr_2O_7$	0.4903 $K_2Cr_2O_7$
N/100 NaOH	0.4000 NaOH

Table V. Initial and Interval Values of pH of Solution

Solution	Days of Submersion at pH					
	0	16	38	51	71	115
N/100 Na_3PO_4	10.68	8.05	6.72	3.48	2.56	2.42
N/100 Na_2HO_4	8.50	5.42	2.91	2.58	2.22	2.25
N/100 NaH_2PO_4	5.75	2.98	2.58	2.45	2.12	2.10
N/100 K_2CrO_4	7.40	7.20	3.71	3.33	2.58	2.86
H_2O	7.00	2.93	2.56	2.45	2.16	2.10
N/100 $K_2Cr_2O_7$	5.25	5.18	3.18	2.98	2.49	2.50
N/100 NaOH	11.50	8.30	6.95	4.09	2.80	2.46

Table VI. Increase in Acidity as Determined by SO_3 Content

Solution	Days of Submersion at SO_3, Ppm			
	38	51	71	115
N/100 Na_3PO_4	324	576	812	1280
N/100 Na_2HPO_4	362	498	792	1180
N/100 NaH PO	356	542	746	1220
N/100 K_2CrO_4	280	508	800	1302
H_2O	324	478	696	1182
N/100 $K_2Cr_2O_7$	306	456	688	1162
N/100 NaOH	298	496	930	1488

Following these preliminary tests more comprehensive tests were designed as follows. Fifty grams of the same sulfuritic material used in the preliminary tests were placed in each of seven 2-liter Erlenmeyer flasks. These samples were submerged in 1000 ml of the solutions in Table IV.

The pH of the solution was determined immediately after it was placed on the sulfuritic material in the flask. The flasks were stoppered wtih cotton to permit free access of air but to keep out dust, and allowed to stand on the laboratory shelves. The pH was taken at intervals and also the SO_3 produced was determined. The initial and interval values of pH are given in Table V.

These data show the progressive time decrease in pH representing increase in acidity, which is determined by the SO_3 content. The SO_3 contents at various intervals are given in Table VI.

Experiment 2 was repeated with sulfuritic material from another source which was ground to −8 +

40 mesh and after washing the HCl was washed with water until the washings failed to give a qualitative test for iron or sulfate. Separate 50-g samples were submerged in 1 liter of each solution in 2-liter Erlenmeyer flasks, stoppered with cotton plugs, and allowed to stand on the laboratory shelves as in the previous experiment. pH values were determined at varying time intervals and SO_4 determinations made. The values for pH and SO_4 at the various times are given in Table VII and VIII.

Table VII. Values for pH and SO_4

Solution	Days of Submersion at pH						
	0	12	24	31	56	78	133
N/100 Na_3PO_4	10.31	7.96	7.40	6.60	3.70	2.91	2.70
N/100 Na_2HPO_4	8.48	6.00	4.05	3.55	2.95	2.42	2.47
N/100 NaH_2PO_4	5.70	3.45	3.20	2.87	2.60	2.18	2.28
N/100 K_2CrO_4	7.10	7.68	7.35	7.04	6.78	4.00	3.18
H_2O	6.40	4.06	3.23	2.98	2.60	2.12	2.15
N/100 $K_2Cr_2O_7$	4.34	6.00	7.87	6.05(?)	5.58	3.32	2.63

Table VIII. Values for pH and SO_4

Solution	Days of Submersion at SO_4, Ppm			
	31	56	78	133
N/100 Na_3PO_4	496	700	1070	1592
N/100 Na_2HPO_4	408	752	972	1492
N/100 NaH_2PO_4	342	618	912	1400
N/100 K_2CrO_4	268	326	572	1200
H_2O	424	724	1010	1316
N/100 $K_2Cr_2O_7$	352	610	902	1532
N/100 NaOH	514	596	998	1810

Table IX. Oxidation Rate for Yellow Pyrite

Solution	Days of Submersion at pH					
	0	16	38	51	71	115
N/100 K_2CrO_4	7.10	7.52	7.30	7.25	7.10	7.14
H_2O	6.70	4.30	4.02	3.75	3.28	3.30

Table X. Oxidation Rate for Yellow Pyrite

Solution	Days of Submersion at SO_4			
	38	51	71	115
N/100 K_2CrO_4	4	14	12	22
H_2O	18	22	38	58

These data are comparable with those of the previous experiment shown in Table III and V.

The difference in oxidation rate of the sulfuritic material associated with the coal measures and pure *yellow* pyrite has been previously demonstrated.*

* Summary Report, *Mine Acid Control*, Mellon Inst., 1954.

However, a comparison of the rate of sulfate formation in water and potassium chromate solution was made in the same manner as the above experiments. Thirty-five grams of $-8+14$ mesh *yellow* pyrite were placed in each of two 2-liter Erlenmeyer flasks. One liter of distilled water was added to one flask and 1 liter of N/100 chromate to the other. The flasks were closed with cotton plugs and allowed to stand on the laboratory shelves. pH and SO_4 determinations were made as for the previous experiment. Data obtained are shown in Table IX and X.

The data in Tables IX and X show the different oxidation rate for yellow pyrite as compared to sulfuritic material from data in Tables V, VI, VII, and VIII. Production of SO_4 in the water solution is somewhat greater than from the chromate solution, which compares with data in Tables V and VII, for the period when the pH of the chromate solution is above 7.0, but when sufficient acid is produced to overcome the alkalinity of the chromate and reduce the pH the rate increases.

A third type of experiment was conducted with the water and potassium chromate solutions. One 50-g sample of the sulfuritic material used in experiment 1 was thoroughly washed with water and placed in an aeration cell, and a second 50-g sample was washed with chromate and placed in a second aeration cell. These cells were so constructed that air could be drawn through the material in the cell and the air subsequently passed through a solution of sodium peroxide to oxidize the issuing SO_2 to SO_4 and retain it in the alkaline solutions. Air was drawn through the samples for a period of 14 days. At the end of this period the sulfuritic material in the cells was washed free of ferrous iron and the washing diluted to 1 liter. The alkaline solutions were neutralized with hydrochloric acid and each diluted to 1 liter. All the samples were analyzed for SO_4 and the washings from the sulfuritic material were also analyzed for iron. Results are shown in Table XI.

These data show a slightly greater reaction, about 5 pct, with the water-washed material. This differential is very small considering the type and state of subdivision of the material. There is also shown a greater amount of SO_4 from the traps, which represents SO_2 produced by the oxidation of the chromate-washed material. During the experiment it was noted that this material became dry earlier than the water-washed material, and since the reaction in the wet and dry state proceeds as follows:

Table XI. Comparative Tests on Two Lots of Sulfuritic Material

Solution on Sulfuritic Material	G. SO_4 in NaOH, Na_2O_2 Trap	Grams SO_4 in Wash Water	Fe in Wash Water	Total M/SO_4/MFe
K_2CrO_4	0.07940	1.0797	0.4914	2.34
H_2O	0.05265	1.2340	0.5551	2.32

Table XII. Comparative Data

Solution	From Table V	From Table VII
Na_3PO_4	0.000222	000240
Na_2HPO_4	0.000200	000224
NaH_2PO_4	0.000212	000215
K_2CrO_4	0.000226	000180
H_2O	0.000201	000198
$K_2Cr_2O_7$	0.000202	000230
NaOH	0.000258	000270

1. (wet) $FeS_2 + 3\frac{1}{2}\,O_2 + H_2O \rightarrow FeSO_4 + H_2SO_4$
2. (dry) $FeS_2 + 3O_2 \rightarrow FeSO_4 + SO_2$

it was anticipated that the chromate cell would produce the greater quantity of SO_2.

Discussion

To evaluate properly the data given above, it is necessary that they be calculated to a common factor. Table XII shows the grams of SO_4 produced per day per gram of sulfuritic material from the two experiments when the sulfuritic material is submerged in the various solutions as taken from the last column of Tables V and VII.

These data indicate a slightly greater rate of reaction in the N/100 NaOH solution, with the N/100 Na_3PO_4 solution next in order, and on an average basis the water solution is the slowest in reaction. However, the differences are probably no greater than the experimental error encountered in using

native material. Thus it may be concluded that the rate of reaction is neither augmented nor inhibited by the presence of phosphates, chromates, or sodium hydroxide in the solution.

During the period of the experiment those solutions containing chromates changed in color from yellow to orange to green indicating the reduction of chromium to the Cr^{+++} state. This change in color did not occur in the chromate in which the *yellow* pyrite was submerged.

Although the pH of the water solution decreased, it did not reach values comparable to the solutions on the sulfuritic refuse. The pH of the K_2CrO_4 was practically unchanged, indicating that the acid produced was used in neutralizing the alkalinity of the chromate by its buffering action.

Experiments in the aeration cells show the increased SO_4 formation caused by aeration in the dry state but do not indicate any inhibiting effect of the chromate deposited on the surface of the sulfuritic material by evaporation of the N/100 K_2CrO_4 remaining on the wet material when it was placed in the cells.

Conclusions

1) Failure to obtain neutralization of acid or evidence of diffusion of NH_3 gas in a mine suggests the futility of attempting to apply inhibitors in a gaseous form to the interior surfaces of an abandoned mine.

2) Phosphates, chromates, or sodium hydroxide have little or no effect upon the rate of oxidation of pyrite or pyritic material either in solution or intimately dried on the surface by evaporation of a solution.

3) The reaction rate of the pyritic material from a coal measure is much greater than the reaction rate of *yellow* pyrite.

4) The reaction rate of sulfuritic waste is greatly increased by aeration in either the wet or dry state.

5) The decrease in pH of buffered solutions on pyritic waste is less rapid than in pure water but ultimately reaches approximately the same value. However, the rate of SO_4 formation is comparable in all solutions.

6) The pH determination is a very poor criterion of reaction rate in buffered solutions, and since the oxidation of the sulfuritic materials is a progressive time reaction, long periods of experimentation are necessary for specific conclusions.

7) There is at present little if any evidence that there is an inhibitor capable of deterring or stopping the reaction between the sulfuritic material associated with the coal measures and the oxygen of the air. If there were such an inhibitor no practical application is known for an abandoned mine.

A Laboratory Method of Determining The Thermodynamic Efficiency of High Explosives

by Leonard L. Felts, George B. Clark, and Joseph J. Yancik

LITTLE information has been published concerning the actual or useful amount of energy obtained from explosives when they are used for blasting. To provide more data on this subject, 8-in. neet cement cubes were blasted in a steel plate box and the breakage energy evaluated by comparison with drop crusher results using Rittinger's,[1, 2] Kick's,[3] and Bond's[4] theories. Theories by Bond and Wang,[5] Lineau,[6] Hatch,[7] Roller,[8] and others[9-11] were not considered applicable. Those used in this investigation may be expressed mathematically as follows:

Rittinger Theory

$$E = K_r \quad \left[\sum_{\infty}^{1} \text{pct}_i \frac{1}{d_i} - \frac{100}{d_o}\right] \qquad [1]$$

Kick Theory

$$E = K_k \quad \left[\sum_{\infty}^{1} \text{pct}_i \log \frac{1}{d_i} + 100 \log d_o\right] \qquad [2]$$

Bond's Third Theory

$$E = K_b \quad \left[\sum_{\infty}^{1} \text{pct}_i \frac{1}{d_i^{1/2}} - \frac{100}{d_i^{1/2}}\right] \qquad [3]$$

where E = energy used in breakage, d_o = average diameters of screen fractions of broken cement particles, pct_i = percent weight of diameter d_i, and $\Sigma\, \text{pct}_i$ = 100 pct.

The above equations were employed to determine values of K_r, K_k, K_b for particle size distributions similar to those obtained from the blasting curves. These values were in turn used as a means of approximating the specific energy of the blasted material. The efficiency of crushing by blasting was then determined by dividing this specific energy by the maximum available work of the explosive per gram of cement.

Knowledge of the failure of solids under *static loads* is developing very rapidly. Their failure under *impact loads* has been investigated and reported in the literature only to a limited extent. Theories of failure[12] under static loading include 1) the maximum stress theory, 2) the maximum strain theory, 3) the theory of constant elastic strain, 4) theory of maximum shearing stress, and 5) Mohr's theory of strength.

All of these have elements of usefulness in establishing an applicable theory for blasting. For highly strained and compressed material such as that surrounding a detonating charge of high explosive, the Mohr theory and its implications appear to be the

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This paper, TP 4213A, is revised from a thesis submitted by L. L. Felts in partial fulfillment of the M.S. degree in Mining Engineering, University of Illinois, 1954. Discussion may be sent (2 copies) to AIME before May 31, 1956. Manuscript, Dec. 27, 1954.

best for further investigation. This is related to triaxial testing[15,16] of rock. In addition to the above, Poncilet[15] has discussed some of the energy aspects of failure by cracking.

According to Taylor[16] the pressures developed by the gases resulting from the detonation of a completely confined high explosive are very high. Calculated values for high NG explosives are in the neighborhood of 1.3x10⁶ lb per in.² Although these pressures are low compared to those that exist in the detonation wave itself, their effects, other than furnishing energy to rupture the enclosing rock, are of first importance.

Little is known about the behavior of solid materials when they are subjected to impact stresses of this magnitude, but the effect of statically applied loads on rocks under high confining pressures has been observed by Griggs.[17]

Although results of these tests may not be directly applicable to the conditions that exist in a rock stressed by an explosion because of time and scale factors, they are significant. The ability of rock to flow and exhibit great strength under high pressure is a substantial argument in favor of the hypothesis that rock so stressed breaks inward from a free face toward the explosive charge rather than outward, rupture also being related to reflected stress waves.

The fracturing of metal plates, rods, and tubes under very high and rapidly applied stresses has been investigated by Rinehart.[18] Explosive charges were used as a means of creating these stresses in order to study the types of fractures that occur, the conditions that lead to these fractures, and the effects that the physical properties of the materials play in the fracturing processes. It was found that under very high rates of loading, metals act as brittle elastic solids.

Explosive Energy. Theory and Testing: Investigations of the physical and chemical phenomena which occur when an explosive is detonated have resulted in the formation of a number of theories. Possibly the most commonly accepted of these is the hydrodynamic theory,[19] which was initially based upon observation of the detonation of gases under controlled conditions.

A number of experimental methods have been designed in the attempt to measure the energy contained in explosives and detonators. They serve,

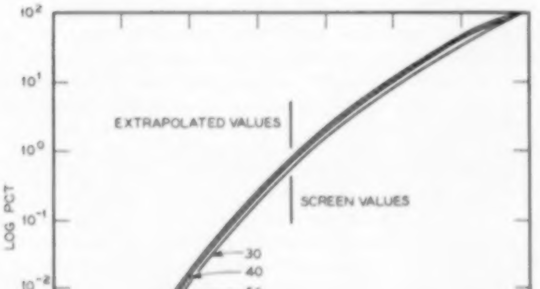

Fig. 1—Cumulative percentage curves for 30, 40, and 50 g of 60 pct dynamite.

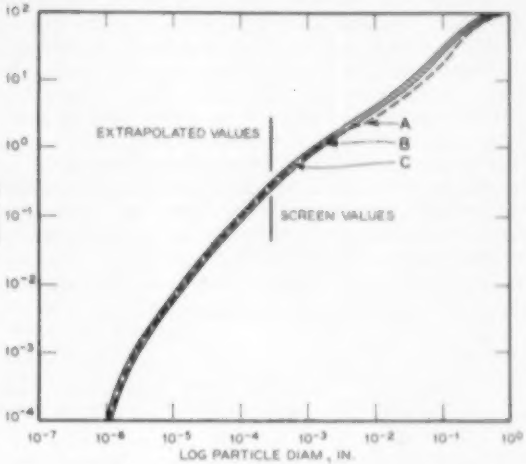

Fig. 2—Cumulative percentage curves for drop crushing tests A, B, and C.

however, only to indicate the comparative energy content above or below that of an accepted standard, and unfortunately may give little indication as to the actual amount of explosive energy that may be usable for blasting purposes.

A detonation represents the maximum rate at which an explosion can take place and is therefore a distinguishing feature of high explosives. Velocity of detonation and the final state of the products are determined by the explosive material, the initial density, and the temperature.

When hydrodynamic treatment is applied to velocity, pressure, density, unit volume, and internal energy per unit mass, detailed chemical processes in the wave front can be largely disregarded, and the laws of conservation of mass, momentum, and energy for the constituents before and behind the wave front can be applied. This theory has permitted formulation of procedures[20] by which the *explosion state* may be calculated, that is, the temperature and pressure which exist in a drillhole immediately after detonation. Properly chosen equations of state also lead to determination of maximum available work, assuming adiabatic expansion of the gaseous products of detonation.

Several types of tests[21] have been devised to measure the effectiveness of high explosives. These include the ballistics pendulum, the ballistics mortar, the nail test, the sand bomb test, and the Trauzl lead block test. In each of these there is no positive means of making a direct quantitative measurement of the strength or efficiency of the explosive. The blasting performance of explosives has been found to correspond to a fair degree with the results from the ballistics mortar, but only approximately.

New Method of Explosive Testing: The method of measuring the efficiency of explosives employed in these tests is simple in basic principle. Solid blocks of neet cement were blasted by known amounts and strengths of explosives and the particle size distributions were employed as bases for calculation of useful energy obtained. These results were then compared with those from blocks broken by a drop crusher where the useful energy input could be very closely approximated.

For three different weights of three different strengths of explosive used in the blasting tests, the particle size distribution and the total energy avail-

able for comminution were determined. Each test was conducted in such a manner that the broken material resulting from the blast could be collected and sized through Tyler standard screens and the Haultain infrasizer.

Thermochemical calculations were made to determine the total energy available from each charge. The total weight of the broken block divided by the total energy available yielded the maximum possible theoretical amount of energy per unit weight of broken material. Only a portion of this total amount is actually used in breakage, however.

Table I. Results of Chemical Calculations

Grams	Dynamite, Pct	Energy, Ft-Lb	Maximum Available Work, Ft-Lb	Portland Cement, G	Total Maximum Available Work, Ft-Lb	Available Work, Ft-Lb Per G
30	30	56,500.6	54,000.1	16,989.1	55,078.4	3.24
40	30	75,334.2	72,000.1	16,104.0	73,078.4	4.54
50	30	94,167.7	90,000.1	16,484.6	91,078.4	5.53
30	60	74,460.2	71,842.1	16,339.4	72,920.4	4.46
40	60	99,280.3	95,789.4	16,307.8	96,867.7	5.94
50	60	124,100.4	119,736.8	17,426.1	120,815.1	6.93
30	90	125,260.7	113,910.0	16,738.9	114,988.7	6.87
40	90	167,031.6	151,880.6	17,021.5	152,958.9	8.99
50	90	208,789.5	189,850.7	16,524.0	190,929.0	11.55
Cap		1,078.3	1,078.3			

To provide as nearly homogeneous material as possible for the blasting tests, blocks were made of straight Portland cement with a minimum amount of water to produce a specimen of maximum strength. The blocks were cured for a minimum period of one month. Inasmuch as a blasting cap was required to detonate each charge, it was planned that enough explosive be used to provide a high degree of energy in comparison with that of the cap. Eight-inch cubes proved satisfactory, as they yielded a wider range of particle sizes for the largest amounts of the strongest explosives. The charges,

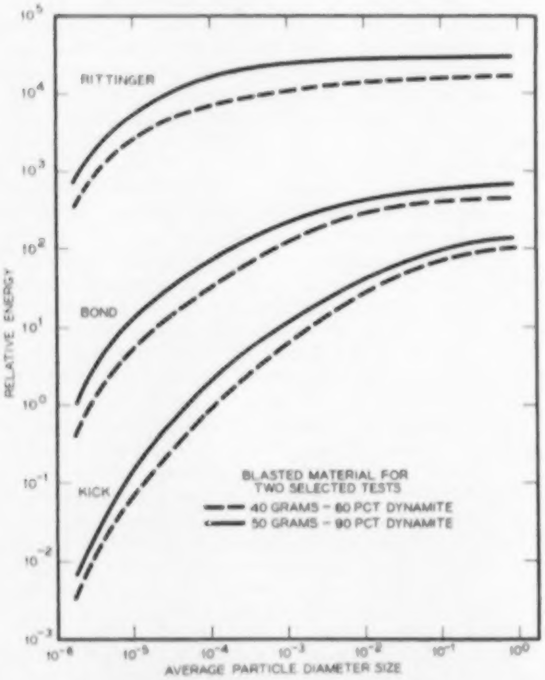

Fig. 3—Cumulative energy.

Dissipation of explosive energy may be classified as follows:

A. Breakage of the test specimen
B. Residual temperature of
   1. Gases of explosion and surrounding air
   2. Broken specimen
   3. Steel box
C. Other work by gases which results in
   1. Deformation of the steel box
      a. Elastic
      b. Permanent
   2. Compaction of the specimen
D. Sound
E. Formation of new chemical products

To determine the amount of explosive energy used in the comminution process, it would be necessary to determine the amount that is lost in B through E above.

complete with the proper percent of paper wrapper and an electric blasting cap, were contained in waterproof cylindrical glass vials. These were cast in the cement blocks, thus providing complete confinement of the explosive.

Explosives used for the tests were 30 pct and 60 pct AN dynamite and 90 pct gelatin dynamite. Charges containing 30, 40, and 50 g of each strength of explosive were used to obtain a series of particle size distributions from the blasted material. The blocks were blasted in a 3-ft reinforced steel box. Loss of broken material was prevented by a steel cover securely bolted to the box, with only a small hole in the center of the cover, through which the blasting cable was attached to the lead wires of the charge.

The broken material resulting from the blast was carefully collected, screened, weighed, and recorded. Minus 200 mesh material was separated into equivalent screen size fractions by processing in an infrasizer.

Total available energy cannot be assumed with accuracy to be total heat of explosion, but must be determined from the laws of adiabatic expansion. Explosion pressure may be determined from a knowledge of the composition and loading density of the explosive, the heats of formation of reactants and products, the chemical equilibria involved, and an equation of state for the gaseous products of detonation. The method employed was similar to Brown's method as modified by Cook. After the *explosion state* has been determined, the gases are assumed to expand adiabatically without change in composition to some final temperature. The heat involved in this charge is assumed to be the *maximum available energy.**

* In the original manuscript the available energy was assumed to be the total heat of explosion at constant volume. Results here are based on the adiabatic expansion law and were calculated by Joseph J. Yancik.

Results of the calculations based on the above procedure were used to determine the available energy per gram of broken material. Final results are expressed in foot pounds per gram of broken material.

Drop Crushing Tests: To evaluate results of the explosive tests, it was necessary to obtain particle size distribution curves similar to those obtained from the screen results of the blasted cubes from a crushing process in which the crushing energy could be measured. The first approach was to crush cylindrical specimens of different diameters and lengths in an attempt to obtain a duplication of the cumulative

curves of blasted material. Test results, however, failed to yield the desired shape of curve. Final design of the drop crusher provided for impact breakage of small cement cubes similar to those used in the blasting tests. The general shape of the curves obtained from these tests differs somewhat in the upper size range but closely approaches that of the explosive curve as the particle size decreases. The apparatus was calibrated to determine the amount of energy used in crushing and the amount absorbed in a manner similar to that employed by Gross and Zimmerly.[26]

The energy required to crush a specimen to any particle size distribution was taken as the total kinetic energy of the drop weight minus the energy absorbed by the apparatus. The absorbed energy for a particular test was found by measuring the deformation of the aluminum cylinder (instead of wires, Ref. 26) and finding the corresponding value of energy from the calibration curve. The crushing energy thus determined divided by the weight of the specimen gave the crushing energy per gram (specific crushing energy) of the broken material.

Results

Many investigations of crushing and grinding energies have employed the assumption that cumulative particle size distribution plots on log-log paper yield straight lines. These lines are extended only to particle sizes as low as 0.7 microns in diam and the particles below this *cut off* size are neglected.

Two assumptions were made in connection with the present work. First, the particle size diameters were taken as the arithmetic mean between two screen sizes. Second, and more important, the plot of the particle size distributions indicated a convex type of curve over the screen and subsieve sizes. The general trend of the curve was followed in the extrapolated portion with the per cent decreasing rapidly as the particle size approached the neighborhood of 300A. Results of data obtained from electron microscope photographs indicated that there were a few particles of this minimum size in the material contained in the last cone of the infrasizer. Data collected by other investigators have also shown that the particle size distribution curve is convex downward in the small particle size range. Since there can be no particles of zero size it follows logically that the curve should drop off rapidly at some minimum particle size. To assume an extrapolated curve of this shape also appears more reasonable than to assume an arbitrary *cut off* size, especially in view of the increase of relative specific energy with decrease in particle size diameters for Rittinger's and Bond's theories.

The particle size fractions below the lowest infrasizer size were taken from the extrapolated curves discussed above for the purpose of determining the values of the *K's* in the equations for the *laws* of crushing. (See Table II and Figs. 1 and 2).

It would appear, on the basis of the results (see Table III) obtained under the test conditions employed and the assumptions made, that explosive breakage is much more efficient than other methods of crushing and grinding. It is realized that for some industrial operations, the end products resulting from a blast may not be similar due to the relatively large amounts of small sized material produced in blasting. Crushing and grinding efficiencies previously reported by other investigators range from a low of less than 1 pct to a high of 17 pct. Schellinger[illegible] found that thermodynamic efficiencies of a ball mill range as high as 17 pct. Inasmuch as the energy losses in ball mill grinding are due to friction of machinery, friction of grinding medium, and probable lower order of efficiency of the ball mill grinding process, the explosive efficiencies calculated herein appear to be in a reasonable range. This is especially true of those determined on the basis of the Rittinger and Bond theories. Results of Schellinger's investigation also appear to justify further the extrapolation of the particle size distribution curves employed in the present investigation. The fact that a significant portion of the kinetic energy of broken particles was converted to breakage energy was doubtless partially responsible for the high efficiencies obtained.

Table II. *K* Values

Drop Crushing Test	Ft-Lb Per G	Rittinger K_r	Kick K_k	Bond K_b
A	1.03	0.000174	0.00680	0.00534
B	1.53	0.000267	0.00975	0.00576
C	1.94	0.000295	0.01188	0.00664

Table III. Explosive Efficiency

Grams	Pct	Rittinger, Pct	Bond, Pct	Kick, Pct
30	30	62.8	66.4	60.0
40	30	66.9	57.5	44.5
50	30	62.4	51.6	38.0
30	60	72.2	59.8	45.7
40	60	66.1	51.3	36.0
50	60	65.2	50.0	32.6
30	90	55.6	42.8	30.4
40	90	53.3	38.0	24.6
50	90	63.6	35.8	20.2

Test results of explosive efficiency range from 20 to 72.2 pct, depending upon the amount and strength of explosive and upon the theory chosen. In the case of the 30-g sample of 30 pct dynamite, the results as plotted in the distribution curve indicated that a portion of the charge may have failed to detonate.

Fig. 3 shows the cumulative distribution of energy for two explosives with respect to particle size. The curves for the Kick theory are more nearly the shape of the cumulative particle size distribution curves than for the other two theories.

Another item of interest is the apparent increase in the *K* values with increased energy input. Only the averages of the results obtained in tests *B* and *C* were used in calculating efficiencies, since the distribution curve (Fig. 2) of test *A* was below those of *B* and *C* in the large particle range and subsequently crossed that of *B* as the particle size decreased. In addition, tests *B* and *C* conform more generally to the blasting tests. The value of the energy *constants* determined from the drop crushing curves is a function of the shape of the curve as well as its position, since the values of all constants were found to increase for curves lying higher on the graph. As most of the blasting curves are above the drop crushing curves, the values of the *K's* for these curves are probably low and likewise the efficiencies calculated therefrom. Calculated efficiencies for the Third theory do, as Bond proposed, fall between those of the Rittinger and Kick theories. It is realized that there is insufficient data available to establish firmly a criteria for blasting efficiencies, and that other tests should be made to confirm the ones found in this investigation.

Sources of Error: There are several possible sources of error in the experimental work described above. Some of these would make the calculated values of the explosive efficiencies too high and some

too low, so that they tend to be compensating to a certain degree.

One of the most obvious of the possible sources of error is the manner in which the particle size distribution was extrapolated for particles below the infrasizer range, i.e., 6.5 microns. For Kick's theory the manner in which the curve is extrapolated has little effect upon the calculated energy values, but for the other two theories it is of considerably increased importance.

One source of error is found in the assumption that the energy absorbed by the equipment is accurately measured by the deformation of aluminum cylinders. This is probably the case for both the determination of the calibration curve and the values measured during actual crushing. However, this means of determining crushing energy does yield a fair approximation.

It was also found that for large crushing energies some of the material was crushed and compacted into a disk more dense than the original material. In all crushing values used as a basis for calculations, therefore, only those specimens were employed which showed very little or no compaction.

The basis of calculating the efficiency of a high explosive is somewhat arbitrary. That is, a reference temperature must be chosen to calculate the available heat. Most thermal values for chemical reactions are referred to 25°C, or room temperature, and the total energy of the explosive used in these experiments is based on this same temperature. If it were possible to refer all calculations to absolute zero, the efficiencies would, of course, be much lower.

One of the greatest difficulties encountered was that of measuring the size fractions of a very fine material. For either the Rittinger or Bond theories, very small amounts of this material represent large expenditures of grinding energy. For more accurate evaluation of the particle size distribution in the submicron range, new and refined techniques must be developed.

Summary and Conclusions

A tentative laboratory method of determining the blasting efficiency of explosives under certain specified conditions was devised and investigated. From this investigation it was found necessary to extrapolate the particle size distribution curves well below the usual assumed *cut off* size. Efficiencies of the blasting process, as determined by this experimental procedure, indicated that the values may be as high as 72 pct when the three different comminution theories are used as bases for calculation. Results obtained by use of the Rittinger and Bond theories are believed to approximate more closely the true efficiencies since the Kick theory gives so little weight to particles in the smaller size range.

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Discussion: Structural and Stratigraphic Control of Ore Deposition in the West Shasta Copper-Zinc District, California

by A. R. Kinkel, Jr.

(*AIME Trans.* February 1955, vol. 201, pp. 165-174.)

Robert T. Walker and Woodville J. Walker (Walker Engineering Corp., Salt Lake City)—Mr. Kinkel's article embodies, in condensed form, the results of the first detailed and complete geological survey that has ever been made of the long neglected but potentially very important West Shasta Copper-Zinc district, in Shasta County, California. In view of the difficulties of the terrain and the complexity of the geological structure, this represents a notable achievement, and it is with regret, therefore, that the writers find it necessary to differ from Mr. Kinkel's conclusions on two important points.

The first has to do with the interpretation of two members of the stratigraphic column of formations in the district; for purposes of comparison, the description of these two members according to Mr. Kinkel is shown parallel to the description of these members according to the writers, see Fig. 14, Mr. Kinkel's nomenclature being used.

The first member, which Mr. Kinkel terms the upper unit of the Balaklala rhyolite, consists of a single, massive, homogenous body of very silicious igneous rock, consisting almost wholly of quartz and albite, and averaging about 80 pct silica. Most geologists of the U. S. Geological Survey who have previously inspected the district have called it an alaskite. It is not a pile of several members, but is a single unit, without internal subdivisions. It has the form of a recumbent lens, approximately flat on the bottom and arched on top, several miles in diameter, and attaining a maximum thickness of 1000 to 1200 ft. It varies in texture from porphyritic to holocrystalline, and contains unusually large quartz phenocrysts up to 4 mm diam. Mr. Kinkel considers this formation to be a thick lava flow that is older than the overlying Kennett shale, while the writers interpret it as being intrusive, constituting a domed sill or laccolith that is younger than the Kennett shale. Their reasons are as follows:

1) This formation shows none of the features which within the writers' experience invariably accompany very silicious lava flows. Nowhere is it vesicular; nowhere does it contain any glass or its devitrified equivalent; nowhere does it contain flow breccia; and nowhere has it a ragged top, in spite of the fact that there is no evidence of surface erosion, which might have removed such a top.

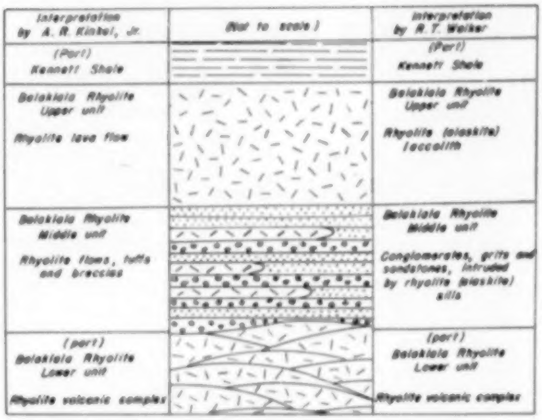


Fig. 14—Partial stratigraphic column as interpreted (left) by A. R. Kinkel, Jr., and (right) by R. T. Walker.

2) Numerous fragments of the overlying Kennett shale, more or less bleached and silicified but with the structure clearly preserved, standing at all attitudes and varying in size from small pieces to good-sized masses, are imbedded as xenoliths in this formation. This could happen only if the Kennett shale were the older, and the alaskitic magma, in the course of invading it, had torn off and engulfed fragments of it.

3) Intervening between the upper surface of this formation and the overlying Kennett shale in many places is a breccia of finely comminuted, bleached, and indurated shale fragments, which must have been created by the heat and pressure of the alaskitic magma in the course of its intrusion. Mr. Kinkel terms this material a tuff, but its constituent fragments are not of igneous material.

4) In places near the contact of this formation and the underlying Kennett shale, the beddings of the shale are distorted and drawn into drag folds.

5) If this formation was a surface flow and the Kennett shale was later deposited upon it, as assumed by Mr. Kinkel, the base of the shale should contain some fragments of this formation, but nowhere along the contact are any such fragments to be seen.

The second member of the startigraphic column, as to whose nature Mr. Kinkel and the writers differ, is the middle unit of the Balaklala rhyolite, which immediately underlies the upper unit previously described. It is of igneous derivation and has the same highly silicious character as the upper unit, but while Mr. Kinkel considers it to be a complex of lava flows and volcanic ejecta, the writers believe it to be largely sedimentary, consisting of conglomerates, grits, and sandstones. The pebbles of the conglomerates vary in shape from ovoid to subangular, while the grits and sandstones are arkosic. Many of the members are well-bedded. The writers consider these beds to be of marine origin, derived by erosion of the top of the underlying lower unit of the Balaklala rhyolite, which appears to be a thick volcanic complex of the same material as that constituting the upper and middle units. When a seat transgresses a land area, as the lower unit appears once to have been, the first sedimentary formation deposited thereon consists of the coarser debris of the previous subaerial disintegration of this land surface, forming what is called a *basal conglomerate*, upon which finer sediments are later deposited. The writers believe this middle unit constitutes the basal conglomerate of the Kennett shale, which once directly overlay it, but from which it is now separated by the laccolith, forming the upper unit, which was intruded along the weak horizon formed by the contact between these two sedimentary formations. The sedimentary beds of the middle unit in places contain interbedded tabular bodies of igneous rock of the same alaskitic composition as the upper unit. Mr. Kinkel considers them to be lava flows, but the writers believe them to be sills, injected at the same time the upper unit was intruded, and derived from the same alaskitic parent magma.

The second point of difference between Mr. Kinkel and the writers has to do with the orebodies of the district. The middle unit of the Balaklala rhyolite, previously described, is the chief ore horizon of the district and contains all the known commercial orebodies. This preference is to be explained by its wide areal extent and continuity; by the permeable nature of the conglomerates and the arkosic grits and sandstones, which chiefly compose it and which are favorable for penetration and travel by mineralizing agencies; and by the almost impermeable roof provided by the massive laccolith constituting the upper unit. Some 15 separate mines have been operated in the district, and Mr. Kinkel appears to consider each one an individual ore deposit, formed by the ascent, along an underlying steeply dipping *feeder fissure*, of mineralizing agencies which, being unable to rise further because of the impenetrable barrier presented by the bottom of the upper unit laccolith, proceeded to spread out laterally in the subjacent middle unit, there depositing their mineral contents to form orebodies—all this despite the fact that extensive mine workings underneath the orebodies have failed to reveal a single *feeder fissure* that can be definitely and positively identified as such. Most of the fissures transecting the orebodies are wholly post-mineral, but where premineral the mineralization dies out with distance below the orebody and the fissures become completely barren, thus strongly suggesting that they were fed from the orebodies and not vice versa. However, until about 25 years ago, this hypothesis of feeder fissures was the standard explanation used by geologists to account for ore deposits of this nature; but since then the fact has become generally recognized and accepted that in such situations mineralizing agencies can and do move laterally for long distances along certain favorable sedimentary beds, forming extensive horizontal or gently inclined sheetlike, ribbonlike, or pipelike orebodies of the type known as *mantos*. The source end of a manto orebody is always some cross-cutting structure, such as a fissure, shear zone or breccia pipe, but it is fed from below only at this point, and throughout most of its course the mineralizing agencies travel lengthwise along the ore horizon, depositing ore as they proceed. Manto orebodies are common in many American and Mexican mining districts, and their habits and peculiarities are now well known.

It is the opinion of the writers that all of the copper-zinc mines of the West Shasta copper-zinc district are located on segments of a few manto orebodies, all of which, from their sources at the southwestern end of the district, trend northeasterly, parallel to each other. Except at their source ends they are confined to the middle unit, in which they occur at two or three separate horizons, the top one just beneath the impervious ceiling provided by the bottom of the upper unit, and the others ranging from 30 ft to 150 ft below. These manto orebodies are physically continuous but not commercially continuous, because they consist of lenticular enlargements, connected by thinner ore, so that while each manto is ribbonlike in general plan it is not of uniform thickness throughout, and only the thicker portions have been mined in the past. Since their formation these manto orebodies have been severed into a number of more or less separated segments by post-mineral faulting, aided by erosion, and these separated segments constitute the different mines previously worked. Fig. 15 provides two diagrammatic longitudinal sections along one of these mantos: the upper section is shown as originally formed, with arrows indicating direction of flow of the mineralizing agencies that deposited it; the lower section is shown according to the present condition after post-mineral faulting and erosion, the black portions representing mined sections

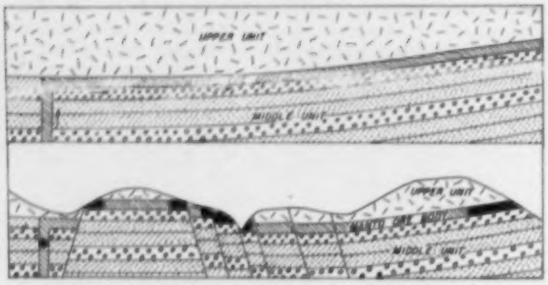


Fig. 15—Diagrammatic longitudinal section (above) along *manto* orebody before faulting and erosion. Arrows indicate direction of flow of mineralizing agencies, which formed the *manto*. Longitudinal section (below) along *manto* orebody after faulting and erosion. Black indicates stoped sections of *manto* orebody.

and the hatched portions those sections which at present are unmined and in most cases unexplored. Few of the mined segments, however, have been completely worked out, and in most of them there still exist one or more faces of commercial ore, beyond which exploration has not yet proceeded. Thus Mr. Kinkel's illustration of the Shasta King orebody (Fig. 9 of his article) is misleading, because one unacquainted with the property would gain the impression from it that the orebody had been delimited and exhausted, whereas actually there is a full face of ore at the north end, beyond which there is more than a mile of virgin ground.

The explanation of this circumstance lies in the fact that the ore of the district is a mixture of iron, copper, and zinc sulphides, which are so finely crystalline and so intimately associated that they cannot be separated by the gravity concentration methods prevailing in the past. Consequently all the ores originally were directly smelted in copper blast furnaces, and when some 30 years ago the smelters were permanently shut down by court injunction because of damage to surrounding vegetation by smelter smoke, there was no alternative at that time to complete suspension of operations, despite the fact that ore reserves were still available. In recent years differential flotation has been developed to a point where it can successfully treat this ore, separating for further treatment a copper concentrate, a zinc concentrate, and an iron concentrate; but the resuscitation of the district by this method has been hindered by destruction of access roads in the creation of the Shasta dam and reservoir, and there has been subsequent delay in constructing replacements.

Mr. Kinkel and the writers, in differing as to the nature of the orebodies, are also at variance as to the best methods for finding new ones and as to the potentialities of the district as a whole. Mr. Kinkel assumes quite correctly that where the middle unit is hidden beneath the overlying upper unit, hitherto undiscovered blind orebodies can exist, but while he believes they can be found by observing certain structural controls such as folds and foliations, he offers no opinion as to exactly where such orebodies may lie or what their possible total value may be. The writers, on the other hand, believing that all unexplored segments of the manto orebodies are potentially commercial and probably comparable in value to those previously mined, are confident that they can predict the existence and location of these hidden manto segments very closely by projecting between known segments, with due allowance for offsetting post-mineral faults; they also think that they can approximate the probable value of such unexplored manto segments.

Thus during the 50 years that have elapsed since the discovery of the district, it has produced about 11,000,000 tons of copper-zinc ore, each ton averaging about 0.028 oz gold, 1.47 oz silver, 3.15 pct copper, and 5.0 pct zinc, most of which was mined over 30 years ago before the district was paralyzed by court injunction. Gross value of this production was about \$125,000,000 at metal prices then prevailing, the zinc not being recovered, and would be over \$350,000,000 under present conditions. The writers estimate that between 75 pct and 80 pct of the volume of the manto orebodies of the district are unexplored and still available, so that the total future production should be between three and four times that of the past, which would place the district among the five top copper mining districts of the U. S.

A. R. Kinkel, Jr. *(author's reply)*—It is unfortunate that in the summary paper on the massive sulfide deposits of Shasta County, California, I was unable to present evidence for many of the statements made there, but this evidence is being presented in full in a forthcoming Professional Paper of the U. S. Geological Survey.

All previous workers in the Shasta area have described the very puzzling and even contradictory field evidence relating to the origin of some of the rhyolitic rocks, and I have no doubt that theories of origin that differ both from my own and from those presented by the Messrs. Walker will be proposed for some of these rocks. Difficulties in interpretation are to be expected in bodies of rock where submarine extrusion of lavas is combined with archipelagic conditions of sedimentation. A few statements in the discussion on my paper should be commented upon here, however, as they are matters of fact, and not of interpretation.

The upper unit of the Balaklala rhyolite does contain tuff and volcanic breccia along the bottom and top of the unit, although it contains a surprisingly small amount of this material, as recognized by Messrs. Robert T. and Woodville J. Walker. Along the top of the upper unit over most of its extent, and extending beyond the limits of the upper unit, are very characteristic varieties of both bedded crystal tuff and shaly tuff. These tuffs contain large quartz phenocrysts and the same igneous material in the matrix as that which is found in the upper unit itself. The tuffs are interbedded with shaly material that grades upward locally into the Kennett shale and shaly tuff without a stratigraphic break. Minor bodies of the coarse-phenocryst rhyolite of the upper unit, however, are intrusive into the rocks below the Kennett, as plugs and dikes that accompanied the extrusion of the main cumulo-dome of coarse-phenocryst rhyolite.

Thin sections and field mapping show that the material at the base of the "laccolith," which the Walkers consider shale that was "indurated" by heat of intrusion, is a silicified shaly tuff in which the silicification is spatially related to ore deposits. That is, over orebodies and in zones of hydrothermal alteration, the tuff along the top of the middle unit has been silicified until it resembles a felsite or hornfels, but away from orebodies the same bed can be traced into normal, unsilicified, rather soft tuff.

At a few localities there are small angular disoriented fragments of shale in a matrix that is a tuff, not an intrusive rock; the tuff matrix is bedded and consists of broken large quartz phenocrysts and subrounded, in part clastic, quartz and albite crystals. This tuff bed commonly shows irregular crumpling, not reflected in the beds above and below, that is caused by syngenetic sliding. The broken shale fragments probably came from thin beds or lenses of shale that were broken during syngenetic sliding. Most of the crumpling, which is a common feature of the shales and tuff at the base of the Kennett formation where this formation overlies the upper unit of the rhyolite, is due to differences in competence between the two rocks and is only developed in areas of folding.

The writer did not intend to give the impression that no ore exists north of the Shasta King mine. The impression is probably due to the small size of the illustration of the mine, in which the northward extent of the ore is left open by question marks. The Messrs. Walker are correct in stating that ore is present along the north face of the Shasta King workings.

aime news

Pacific Northwest Regional Conference to be Held in Seattle in May

The AIME Pacific Northwest Regional Metals and Minerals Conference will be held on May 3, 4, and 5 in Seattle.

Earl R. Marble, general chairman of the conference, has announced that Grover J. Holt, AIME President-Elect, and general manager, Ore Mining Dept., Cleveland-Cliffs Iron Co., Ishpeming, Mich., will be the featured speaker at the banquet on Friday. The speaker at the Mining Branch luncheon on Friday will be Desmond F. Kidd, past president of the Canadian Institute of Mining and Metallurgy, and president and general manager, Attwood Copper Mines Ltd., Vancouver, B. C. The Metals Branch luncheon will be held on Thursday. The speaker will be J. Gordon Parr, Associate Professor of Metallurgy, University of Alberta, Edmonton. The technical program will feature papers of interest to all sections of the AIME. Assistant general chairman of the conference is Marshall Huntting.

GROVER J. HOLT

Mining Branch

The Co-Chairmen for the Mining Branch technical sessions will be: W. C. Douglass, vice president, Kelowna Mines-Hedley Ltd., Seattle, and H. C. Hughes, chief inspector of mines, Dept. of Mines, Victoria, B. C. Mr. Douglass will also serve as chairman of the Mining Branch activities.

The following Mining papers will be presented: *Petroleum and Natural Gas in British Columbia, Importance of Washington Coal Reserves in the Regional Power Picture, Rock Bolting in Metal Mines*, and *Underground Transportation at the Pend Oreille Mine*.

One of the papers at the Minerals Beneficiation session will be *Sunshine Milling Practice*. This will include a report on work being done by the Sunshine Mining Co. on electrolytic antimony. For the uranium men there will be *A Review of Uranium Milling Practice to Date*.

This session will also include: *Factors Influencing the Selective Flotation of Complex Lead-Zinc Ores*, and *The Impact of New Flocculents on Hydrometallurgical Processes*.

The Geology Subdivision will present *Tectonic Patterns of the Western United States, Granites of Magmatic Origin as Distinguished from Granites of Metasomatic Origin*, and *Localization of Ore in the Kootenay Arc*. There will also be a paper correlating expanded rates of ore discovery with geological field studies in the Northwest.

The Industrial Minerals Div. will include *Industrial Mineral Industries of the Pacific Rim—Recent Developments and Expected Trends; Recent Advances in the Structural Clay Products Industry;* a paper on the requirements for industrial silica; and one describing a glass-container plant.

A Mineral Industries Education Div. session has been scheduled by Drury A. Pifer, director, School of Mineral Engineering, University of Washington, Seattle, for 8:00 p.m. Thursday. Among the topics to be covered are *Problems in High School Education, Freshman Engineering Education*, and *Trends in Engineering Education*.

Metals Branch

The Chairman of the Metals Branch technical sessions will be J. G. Johnston, chief metallurgist, Bethlehem Pacific Coast Steel Corp., Seattle. He will also be chairman for Metals Branch activities at the conference.

Demonstrations and Tours

X-Ray diffraction equipment demonstrations will be given all day Friday and until noon Saturday at Milnor Roberts Hall, on the University of Washington campus. Tours through several interesting industrial plants in Seattle, Renton, and Tacoma are being arranged for Saturday morning.

The chairmen of the various branch activities are: J. G. Johnston, Bethlehem Pacific Coast Steel Corp., Seattle, Metals Branch; W. C. Douglas, mining engineer consultant, Seattle, Mining Branch; and D. A. Pifer, School of Mineral Engineering, University of Washington, Minerals Industry Education. Working as division chairmen are: H. C. Hughes, Dept. of Mines, Victoria, B. C., *Mining in British Columbia*; Paul Billingsley, mining geology consultant, Burton, Wash., *Geology;* W. D. Nesbeitt, Allis-Chalmers Mfg. Co., Spokane, *Minerals Beneficiation;* H. D. Kelley, U. S. Bureau of Mines, Seattle, *Industrial Minerals;* Lloyd Banning, U. S. Bureau of Mines, Albany, Ore., *Iron and Steel;* E. C. Roberts, University of Washington, Seattle, *Physical Metallurgy;* and R. E. Shinkoskey, American Smelting & Refining Co., Tacoma, Wash., *Extractive Metallurgy*.

Rocky Mountain Minerals Conference

The 3rd Rocky Mountain Minerals Conference will be held September 26, 27, and 28 at the Newhouse Hotel, Salt Lake City.

R. C. Cole, general chairman for the conference, will be assisted by A. J. Thuli, Jr.

Technical papers will be presented from all branches of the Institute and several interesting field trips are planned. Special attention will be given to the ladies' program and there will be the usual round of social activities.

The Salt Lake meeting will precede the American Mining Congress Convention in Los Angeles, making it possible for members to attend both events.

E. J. Longyear Fund

The Edmund J. Longyear Memorial Fund of $3400 has been established through the University of Minnesota to provide one or more scholarships for training in mining and metallurgical engineering, geology, and other basic earth sciences during the school year 1956 to 1957. The scholarships will be administered by the Bureau of Student Loans and Scholarships and will be supported annually by E. J. Longyear Co.

Around the Sections

• The **Chicago Section** met February 1 at the Chicago Bar Assn. The presiding officer was Paul R. Nichols and 49 attended. Dana W. Smith, associate director and laboratory manager, metallurgical research dept., Kaiser Aluminum & Chemical Corp., spoke on "Recent Developments in the Aluminum Industry."

• The **Montana Section** met January 25 at the Physics Auditorium, Montana School of Mines, Butte. J. R. Van Pelt presided. After a short business session, V. D. O'Leary, superintendent, Mt. Con Mine, Butte, gave an interesting paper entitled "Hydrolic Slime Filling at the Mt. Con Mine." This was followed by a lively discussion. According to report the Section went on record to back Roger Pierce for a directorship in the AIME.

• The **Reno, Nev., Subsection** met on January 13 at El Cortez Hotel. E. L. Stephenson was chairman and 40 attended. Joseph Lintz, Jr., secretary, Nevada Oil & Gas Conservation Commission presented "The Occurrence of Oil in Railroad Valley, Nevada." He discussed the geology of the several producing wells of the area and gave the history of production of each. Mr. Lintz said that in three of the producing wells the oil horizon was in a different formation. Officers for 1956 are: E. L. Stephenson, Chairman; A. L. Engel, Vice Chairman; and Henry P. Ehrlinger, III, Secretary. Erich J. Schrader was the guest of honor at the meeting on February 10. Mr. Schrader, who became a Legion of Honor Member this year, related some of his experiences with the AIME and the mining profession. Plastic name tags have been secured for the membership of the Reno Subsection.

• The **Washington, D. C., Section** met January 10 at the Cosmos Club. H. DeWitt Smith, chairman, O'okiep Copper Co. Ltd., was the guest of honor. Mr. Smith, who was 1955 President of the AIME, discussed Institute affairs. On February 17 the Section met at the National Press Club. Frank M. Stephens, Jr., discussed "Research and the Future of Extractive Metallurgy."

• The **Lima, Peru, Section** held a luncheon meeting on December 21 in the America Room, Hotel Bolivar. Elynor Rudnick, president, Helicopter Assn. of America, was the speaker. Her subject was "New Horizons For The Helicopter." Miss Rudnick illustrated her talk with a 20-min film. The luncheon was attended by 67 engineers and guests.

• The **Carlsbad Potash Section** met on January 13 at the Riverside Country Club, Carlsbad, N. M. H. N. Clark presided and 58 were present. C. R. Kuzell, vice president, Phelps Dodge Corp., and a Director of the AIME, spoke on AIME organization and pending legislation. After his talk, Mr. Kuzell showed a film on Arizona resources. R. E. O'Brien AIME Field Secretary, was among those present.

• The **Southwestern New Mexico Section** held a technical session on November 17 at the Bayard Lions Club Bldg., Bayard, N. M. Chairman E. A. Slover presided and 34 were present. The principal speaker was Clay T. Smith, professor of geology, New Mexico Institute of Mining and Technology, Socorro. Mr. Smith gave a very interesting and instructive discussion on "Uranium in New Mexico."

• The **Colorado Section** held a dinner meeting on January 19 at the University Club, Golden, Colo. Max Gelwix of the planning dept., Climax Molybdenum Co., presented a review of the functions of the planning or research dept. at Climax, and of the results obtained from intensive study of high cost items such as drill steel, and ball mill liners.

• The **Ajo Subsection** met January 12 at the Copper Coffee Shop. D. R. Cratty was the presiding officer and 23 attended. The speaker was J. W. Byrkit, Smelter superintendent, Phelps Dodge Corp., New Cornelia Branch. After briefing members on the flowsheet of the Ajo Smelter, Mr. Byrkit presented a paper "The Ajo Holding and Oxidizing Furnace."

The Executive Council of the Utah Section, Student Advisors, and faculty members were among the 87 attending the dinner for the University of Utah Student Chapter by the Section on January 19.

• The **Utah Section** met January 19 at Newhouse Hotel, Salt Lake City. R. L. Done, manager, Atomic Energy Div., Phillips Petroleum Co., Idaho Falls, Idaho, was the speaker. His subject was "Utilization of Uranium in Nuclear Reactors." The meeting was preceded by a dinner given for the University of Utah Student Chapter. D. F. McElhatten and K. C. Olsen were in charge of all arrangements.

• The **Alaska Section** has announced that the following will hold office until October 1956: Douglas B. Colp, Chairman; Ted A. Loftus, Vice Chairman; Gil Monroe, Secretary-Treasurer; Donald J. Cook, Membership Chairman; Earl H. Beistline, Student Relations; Donald MacDonald III, Program Chairman; Jack R. Hoskins, Reporter. Members of the Executive Committee are: Douglas Colp, Ted A. Loftus, Gil Monroe, Jack R. Hoskins, and James Crawford.

The **St. Louis Section** held a joint meeting with the AIChE St. Louis Section on January 20 at the York Hotel. Presiding officers were A. W. Schlechten, for the AIME, and A. T. Pickens, for the AIChE. There were 53 AIME members and 44 AIChE members present. The speaker was F. A. Forward, head of the mining and metallurgy dept., University of British Columbia. Mr. Forward spoke on the developments in pressure leaching of copper-nickel ores with ammonia, and both basic and acid leaching of uranium ores. He also covered the laboratory development of the processes, pilot plant operation, and the design and operation of full-scale plants.

• The **New York Section** met January 19 at the Mining Club. Taconite mining and processing in Minnesota was illustrated by a film. A spokesman from the Reserve Mining Co. was present to answer questions.

29th Annual Meeting of the Minnesota Section Held in Duluth

The Minnesota Section held its 29th Annual Meeting on January 9 at the Shrine Auditorium, Duluth, in conjunction with the 17th Annual Mining Symposium sponsored by the University of Minnesota Center for Continuation Study held on January 10 and 11.

The AIME meeting started early on Monday with more than 440 attending.

At the first technical session, with John D. Boentje, Jr., presiding, Frank Werther of Pickands, Mather & Co., gave a talk on "Soils Solidification in Tunnel Driving." Mr. Werther was followed by Gilbert Wrenn of the University of Minnesota. His topic was "Human Relations."

Ralph Marsden of Oliver Iron Mining Div., U. S. Steel Corp., introduced the speaker, H. R. Steacy, Technical Officer in Charge of Radiometric Laboratory, Canadian Geological Survey, Ottawa, at the luncheon held in the Hotel Duluth. Mr. Steacy gave an address on "Uranium in Canada," prepared by A. H. Lang of the Canadian Geological Survey. Mr. Steacy outlined the exploration program in Canada, covering the period when it was executed by the Government, and the shift to private exploration. He gave a brief history of the great Canadian uranium deposits and producing areas including Eldorado, Gunnar, and Blind River, and forecast that by 1958 uranium would be a $180 million industry in Canada. He also underscored the high rate of failure in prospecting. Even with modern skill and equipment a thousand leads and showings must be followed before the one economically feasible deposit is found.

Section Chairman Milton F. Williams looks over the registration desk during the meeting. Seated at Williams right is L. A. Thorsen, and at his left, Howard Evans, energetic chairman of the Section membership committee.

The afternoon session, with Kenneth L. Merklin presiding, began with a film entitled *Mining for Nickel*, which covered mining methods at various underground and open pit mines by International Nickel Co. After the movie, Jay Y. Welsh and B. R. Babbitt of Manganese Chemical Co., presented a paper "Manganese from Domestic Ores by the Ammonium Carbonate Process." The final paper was "Sintering of Iron Ores and Concentrates at the Extaca Plant" given by R. L. Bennett, R. E. Hagen, and M. V. Mielke of the Oliver Iron Mining Div., U. S. Steel Corp.

In the evening the Minnesota Section held its Annual Banquet at the Hotel Duluth. Milton F. Williams was toastmaster, and Elmer W. Pehrson, chief, Div. of Foreign Activities, U. S. Bureau of Mines, was the principal speaker. Mr. Pehrson's topic was "National Security vs Mineral Imports." Grover J. Holt, AIME President-Elect, spoke briefly on behalf of the national AIME organization.

The symposium opened on Tuesday, with B. A. Andreas presiding. The program covered various aspects of mining, including drilling blasting, stockpiling, and research.

On Wednesday the lead-off panel discussion was the center of interest. Donald W. Scott was moderator. Later there was a discussion on beneficiation research problems.

When leaving New York in the middle of January to go to Duluth, we were asked: Who would go to a meeting in Duluth at this time of year? The answer: Anyone who has an interest in iron ore, its mining, or its handling! At that time the temperature at Idlewild was 25°—at Duluth it was 26° *above*.

H. R. Steacy of the Canadian Geological Survey was the speaker at the luncheon Monday. Among the guests of honor was President-Elect Grover J. Holt, at the extreme left in the picture.

Engineering Societies Back Biggest Nuclear Conclave

AIME Makes Major Contribution to Program

Cutting squarely across the chosen areas of the nine member bodies of Engineers Joint Council and 17 affiliated groups more than 223 technical papers presented the greatest aggregate of nuclear technology yet heard in this country, in Cleveland, December 12-16, at the Nuclear Engineering and Science Congress. Simultaneously the 1955 International Atomic Exposition displayed the wares of 176 commercial enterprises (16 foreign) to 14,000 scientific, engineering and casual visitors including college and high school students from which the next generation of nuclear scientists and engineers must be recruited.

Technical Sessions

Some 23 papers were presented under the sponsorship of AIME. In addition, to these papers on the metal physics aspects of Nuclear Metallurgy, AIME papers were also given on the extractive metallurgy of uranium ores. It is planned to publish several of these papers in the May issue of JOURNAL OF METALS.

Most of the papers presented at the meeting were preprinted at cost for the Congress by the American Institute of Chemical Engineers, and were sold at the Congress for 30¢ per preprint. Preprints can be obtained from AIChE, 25 West 45th St., New York 36, N. Y.

Working with the Engineers Joint Council Program Committee in obtaining the AIME papers were AIME members, D. W. Lillie, D. H. Gurinsky, Frank Foote, and W. R. Opie.

Heading the list of special events was the all-Congress Dinner on Wednesday, December 14. Speaker for the occasion was Admiral Lewis Strauss, Chairman of the Atomic Energy Commission. After reading a special message from President Eisenhower to the Nuclear Congress, Mr. Strauss described the possibility of forfeiting our newly found treasure, unless we are able to develop a reservoir of scientific and engineering talent to carry on nuclear research. With only some 500 scientific graduates trained in nuclear technology per year and a need three or four times greater, we are losing the race to the Soviet Union.

A Conference for Management, sponsored by the Cleveland Engineering Society, was held on Thursday evening. Many of the cooperating societies held special meetings, luncheons, or dinners. The Cleveland Section of AIME sponsored a luncheon on Thursday, December 15, for members attending the Nuclear Congress. Harry B. Osborn of Ohio Crankshaft Co. served as Toastmaster and Secretary Ernest Kirkendall spoke briefly on present AIME plans.

Chairman of the General Committee sponsoring the Nuclear Congress was Dean John R. Dunning of Columbia University. AIME representative on this Committee was J. H. Frye, Jr. of OakRidge National Laboratory. The Program Committee was headed by an AIME member, Donald L. Katz, Chairman of the Dept. of Chemical and Metallurgical Engineering, University of Michigan. Chairman of the Publications Committee was Stanley Tucker, Advertising Director of AIME. Much of the Credit for the success of the Nuclear Congress goes to hard-working F. Paul Lange, Secretary of the Engineers Joint Council, who coordinated the arrangements made by the 26 cooperating societies. Local arrangements were handled by the Cleveland Engineering Society.

MINING BRANCH MEMBERSHIP DRIVE

The results of our membership campaign in 1955 are gratifying and we have been able to strengthen the Institute and revitalize it by obtaining Student Members and Junior Members in larger number than heretofore. Growth of approximately 20 pct in the last two years is proof that when all of the members of our Institute apply themselves to a purpose, they succeed.

We need each and every one of the members of the Institute to cast about them for people interested in the extractive industries, not only as engineers, but in associated functions and responsibilities. The AIME is not entirely an engineer's society. Roughly 25 pct of our full Members, and more than 30 pct of our Associate Members are not graduate engineers. They are people interested and convinced of the purpose of our Institute.

Many of you, in response to our President's request for help in this matter, have submitted names of candidates for the Institute. This has now become an annual custom, but we would very much appreciate your help and cooperation throughout the whole of the year.

It is hoped that certain plans of the National Membership Committee to help the Sections in obtaining new members will bear fruit in 1956. But the best of plans can only be as good as the conviction of each and every member that our field should be broadened and strengthened.

It is our hope that we will be able, from time to time, to report our progress to the membership at large. At this time of the year we need your help not only in securing new members but in convincing those members who are about to let their membership lapse for nonpayment of dues that they are making a mistake. Your continued participation in our activities is convincing proof that the Institute has something to offer to every individual.

A new group of officers of the National Membership Committee has now taken over and is faced with a record difficult to equal or surpass. Last year the Institute membership was the highest ever. We promise to do everything within our power to further our aim to make the AIME the professional society of all those interested in the minerals and extractive industries whatever the specialization or the field. AIME has something to offer to candidates, we have very much to give to our members, and as your new officers of the National Membership Committee, we hope that you will help us in our campaign to add to the strength and power of the Institute.

*C. E. Golson**

** Chairman, AIME National Membership Committee*

PERSONALS

FRED LOHSE

Fred Lohse has been appointed technical assistant to the manager, Chemicals Div., Kaiser Aluminum & Chemical Corp., Oakland, Calif. Mr. Lohse has had more than 20 years' experience in chemical engineering and related fields. He joined the Kaiser industries in 1939 as a process engineer for Permanente Cement Co. Two years later he was appointed project engineer for the Moss Landing, Calif., sea water magnesia plant. Mr. Lohse has also been assistant manager, Development & Engineering Div., Henry J. Kaiser Co., Oakland, and assistant manager, Firelands Div., Permanente Metals Corp., Marion, Ohio. Mr. Lohse received a B.S. in chemistry from the University of Nevada in 1930 and later completed two years of graduate studies in chemical engineering at the University of Illinois.

Robert H. Chisholm has resigned from Pacific Isle Mining Co., Hibbing, Minn., and accepted a position as assistant to the president, Carbon Limestone Co., in Ohio.

Hugh C. Morris is a geologist at the Sullivan mine, Canadian Mining & Smelting Co., Kimberley, B. C.

H. Marks is now a consultant mining engineer with headquarters in Sydney, Australia. His address is: Box 3751 G.P.O., Sydney, N. S. W. Mr. Marks resigned from the Mines Dept, Brisbane, Queensland, early last year and for six months was field engineer with Uranium Holdings N.L. operating in the Northern Territory.

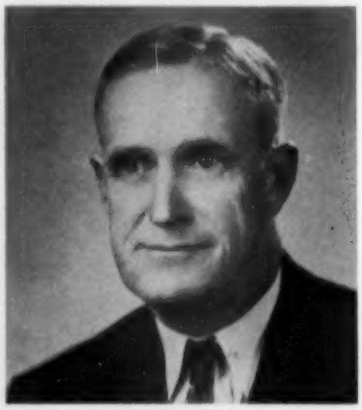
John D. Rankin has been promoted to mine superintendent, Cía. Minera Condoroma S.A., a branch of Cía. Minas del Peru S. A.

Charles H. Lambur is president, Tekera International Inc., New York. Mr. Lambur was with U. S. Collieries Inc., New York.

E. T. Casler has been promoted to manager, Florida operations, Phosphate Minerals Div., International Minerals & Chemical Corp. Mr. Casler joined International in 1942. After a year as staff engineer for the Phosphate Div., he was appointed assistant manager, Florida phosphate dept. Mr. Casler graduated from the University of Florida in 1913, and in 1944 he received the honorary degree of doctor of science from that university in recognition of his contributions to the progress of phosphate mining in Florida. Mr. Casler replaces **Floyd B. Bowen** who was promoted to production manager for the Phosphate Minerals Div. operations in Florida, Tennessee, and Montana, about a year ago.

Merritt K. Ruddock is with Almar Exploration Co., Salt Lake City. Mr. Ruddock was at Cal Uranium Co., Moab, Utah.

Robert L. Haffner, Eagle-Picher Co., Miami, Ariz., has been promoted to general superintendent, and transferred to the company's Illinois-Wisconsin operations.



L. F. PETT

L. F. Pett, general manager, Utah Copper Div., Kennecott Copper Corp., Salt Lake City, has been elected president of the Utah Mining Assn. for the year 1956. **Clark L. Wilson,** vice president, New Park Mining Co., Salt Lake City, is first vice president, and **Charles A. Steen,** president, Utex Exploration Co. Inc., is second vice president. The following were re-elected: **A. G. Mackenzie,** as vice president and consultant; **Miles P. Romney,** as secretary-manager; and **Walter M. Horne,** assistant secretary-manager.

Al Gobus of Research & Control Instruments Div., North American Philips Co. Inc., Mount Vernon, N. Y., has been assigned to assist with technical problems at the company's new office in Newark, N. J. Mr. Gobus will deal with X-ray radiographic units.

H. C. McCollum, vice president, Peabody Coal Co., Taylorville, Ill., has moved to the company's office in St. Louis.

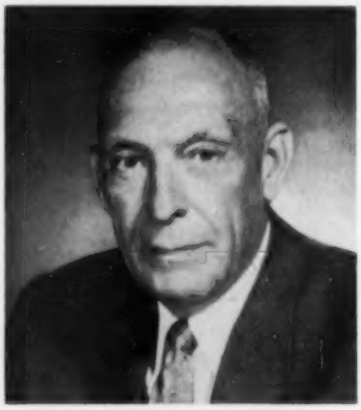
Allen B. Hollett is with Lake Asbestos of Quebec Ltd., a subsidiary of the American Smelting & Refining Co., near Thetford, Que. Mr. Hollett has been mining engineer for Asarco in New York since 1952.

Leo D. Shelledy has taken a position as sales engineer with Schloss & Shubart, Denver. Mr. Shelledy was mining engineer and sales engineer, Joy Mfg. Co., Salt Lake City.

Ivor G. Pickering, who was assistant mechanical engineer, Kennecott Copper Corp., has been appointed to the newly created position of chief designing engineer at Kennecott's Western Mining Div. engineering dept., University of Utah. Mr. Pickering has worked in the mining field since 1935 when he joined Utah Copper Co. as a trackman at Bingham Canyon. He held various positions at the Bingham Canyon operations and the mills dept. of the Utah Copper Div. In 1946 Mr. Pickering became associate engineer, Union Oil Co., Los Angeles. He rejoined Utah Copper Div. and became first designer in the mechanical engineering dept. of the Arthur mills in 1949. Mr. Pickering graduated from the University of Utah with a B.S. in metallurgical engineering in 1939.

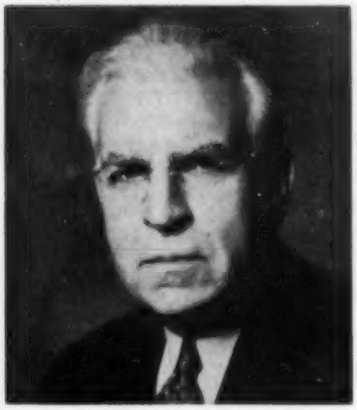
Lewis E. Scott, geologist, construction dept., Oregon State Highway Commission, Salem, Ore., is with the U. S. Bureau of Public Roads, San José, Costa Rica.

Allan E. Jones has been appointed manager, Atomic Energy Commission, Grand Junction Operations Office, Colo. Mr. Jones has been deputy manager of the office since September 1955.



JOSEPH W. BARKER

Joseph Warren Barker, chairman of the board and president Research Corp., New York, is president of the American Society of Mechanical Engineers for the year 1955 to 1956. Mr. Barker assumed office at the Society's Diamond Jubilee Annual Meeting in Chicago last November. He was presented to the members by **David W. R. Morgan,** retiring president of the Society.



L. C. CAMPBELL

L. C. Campbell, formerly vice president in charge of mine operations, Coal Div., Eastern Gas & Fuel Associates, Pittsburgh, is now responsible for the firm's long-range development of coal properties. He will serve as the company's representative in the Bituminous Coal Operators Assn. and handle special assignments for the president of the company. Mr. Campbell has been associated with coal operations of Eastern Gas & Fuel Associates and its predecessors for more than 25 years and has been in charge of the company's mining operations since 1941. He is president of the National Coal Assn.

Lowry M. Wilson has joined Gulf Minerals Co., Salt Lake City. Mr. Wilson was with American Smelting & Refining Co., Salt Lake City.

E. C. Roper has been elected executive vice president, Howe Sound Co., New York. Mr. Roper was manager, Britannia Mining & Smelting Co. Ltd., Britannia Beach, B. C. He will supervise all the operations of Howe Sound Co. and its subsidiaries in the U. S., Canada, and Mexico.

Neil O. Johnson has been appointed plant manager, Foote Mineral Co., Kings Mountain, N. C. Since 1939 Mr. Johnson has been technical representative in the explosives dept., E. I. du Pont de Nemours & Co., Wilmington, Del. In World War II he was a major in the Corps of Engineers, and served in Alaska on the Alcan highway project. Later he became a project officer on the engineering Board of the Joint Army-Navy Board and served as a consultant on advanced design equipment. For some years Mr. Johnson was associated with the Hog Mountain Gold Mining & Milling Co., Birmingham. He graduated with an E.M. from the Colorado School of Mines in 1933.

Robert M. Hardy, Jr., is executive vice president, Sunshine Mining Co., Spokane. Mr. Hardy was assistant to the general manager at Kellogg, Idaho.

Edward M. Tittmann, general manager, Western dept., American Smelting & Refining Co., Salt Lake City, has been appointed president, Southern Peru Copper Co., which is now developing the Toquepala deposits in Peru. Mr. Tittmann is succeeded at Salt Lake City by **W. G. Rouillard,** manager, Tacoma smelter. **Ralph L. Hennebach** of New York is assistant general manager, Western dept., Salt Lake City. **R. M. McGeorge,** zinc plant superintendent, Selby, Calif., is ore buyer and assistant to the manager, Garfield smelter.

J. K. Richardson, assistant general manager, Chino Mines Div., Kennecott Copper Corp., Hurley, N. M., has been elected first vice president, New Mexico Mining Assn.

W. J. Rundle, formerly associate professor of mining and metallurgy, University of Wisconsin, Madison, is with Cyprus Mines Corp., Los Angeles.

Erwin C. Winterhalder, is now geologist with the Atomic Energy Commission, Exploration Div., Grand Junction, Colo.

Walter N. Campbell, mine manager, Thomas Consolidated Mines Inc., Spokane, is geologist and mine manager, Mernab, Moab, Utah.

H. C. Meyer, chairman of the board, Foote Mineral Co., Philadelphia, has retired. Mr. Meyer will continue to assist the company in an advisory capacity. He joined Foote Mineral Co. as a mineralogist in 1906 and was president of the company from 1936 to 1952. During this time Mr. Meyer traveled extensively through the U. S., Europe, and South America collecting mineral specimens. He is one of the country's leading authorities on commercial mineral deposits.

L. A. Norman, Jr., has joined the staff of Equipment Engineers Inc., San Francisco. Mr. Norman was with the California Div. of Mines.

Robert B. Brackin, formerly beneficiation plant superintendent, Kaiser Steel Corp., Eagle Mountain, Calif., is senior engineer, Engineering Service Div., E. I. du Pont de Nemours & Co., Wilmington, Del.

Carl S. Westerberg has joined Heiner Coal Co., Salt Lake City. Mr. Westerberg was formerly with Link-Belt Co., Chicago.

Daniel B. Hurley, chief exploration engineer, Grand Junction Exploration Branch, Atomic Energy Commission, is manager, uranium dept., Tidewater Associated Oil Co., Albuquerque, N. M.

Wallace G. Woolf, manager, electrolytic zinc plant, Sullivan Mining Co., Kellogg, Idaho, has been appointed manager of the Bunker Hill & Sullivan Mining & Concentrating Co.

Clifford C. Furnas, chancellor, University of Buffalo, has been appointed Assistant Secretary of Defense, Research and Development, by **President Eisenhower**. Mr. Furnas has been serving the Dept. of Defense Research and Development organization in a consultant capacity and has been associated with its predecessor organization, the Research and Development Board, in various capacities since 1948. From 1946 to 1955 Mr. Furnas was director of the Cornell Aeronautical Laboratory, Buffalo. He graduated from Purdue University in 1922 and received a Ph.D. from the University of Michigan in 1926. In addition to being the author of a large number of technical articles, Mr. Furnas has also written several books, including *The Next Hundred Years* and *America's Tomorrow*.

William H. McNair has gone into partnership with **Willard A. Houser** for the purpose of minerals exploration at 436 Crocker Road, Sacramento 21, Calif. They are using a helicopter in the search for uranium. Mr. McNair was a geologist with The Anaconda Co., Spokane, and had been with the company for six years.

Melvin Carlson of Billings, Mont., is the recipient of a Mineral Engineering Fellowship at the Harvard Graduate School of Business Administration. Mr. Carlson graduated from the Colorado School of Mines with a B.S. in mining engineering in 1950.

John W. Taber is general superintendent, Cummings-Roberts Fluorspar Operations, Darby, Mont. Mr. Taber was with the U. S. Bureau of Mines, Spokane.

Donald McKechnie has been appointed superintendent, Franklin-Sterling mining operation, New Jersey Zinc Co. in New Jersey. Mr. McKechnie, formerly assistant superintendent, has been employed at the Franklin-Sterling operation for more than 36 years. **Sterling S. Huyett** has been appointed superintendent of the Friedensville operation, a mine in the process of development in Pennsylvania. Mr. Huyett was assistant to the general superintendent. He started his employment at the Gilman, Colo., mine in 1945 and later was superintendent, Hanover, N. M., mine.

Donald A. Smith is now with Mindanao Mother Lode Mines Inc., Cabangan, Zambales, P. I. He was with Lepanto Consolidated Mining Co.

John D. Crickmore has joined Harry Hayley & Sons Ltd. of Ottawa, as plant engineer and director of test and design laboratory. Mr. Crickmore was formerly a graduate student at Stanford University. He received an M.S. in mining engineering last June.

OBITUARIES

Burt Boynton Brewster

An Appreciation by

J. Fred Johnson

Burt Brewster (Member 1929) was born in 1890 in Brussels, Belgium, a 12th generation direct male descendent of Elder William Brewster, passenger on the Mayflower and a principal in finding Plymouth Rock. As a Yankee he attended the traditional elementary and preparatory schools but, as a mining engineer, he matriculated at the Michigan College of Mines.

His father being an important factor in the coal mining business and an erstwhile friend of John L. Lewis, Burt naturally gravitated to the coal end of mining and was an authority on coal equipment for the Sullivan Machinery Co. for his 20 years with that company. On one of his diversions to metal mining I met Burt in 1920, on the 1940 level of the Chief Consolidated mine where there were four men and three pumps at the bottom of a winze, and a bulky piston hoist at the top that was frequently slipping its valves and fouling the whole deal. Burt set up a bar, mounted one of his turbinair hoists on it alone, and proceeded to hoist at about twice the speed and with no delays. I have been a friend of Burt's and of the turbinair hoist ever since. One particular incident always will be in my memory. Hugh Conelly, master mechanic at the Chief, made a slighting remark about some piece of Sullivan machinery so there was a fight on right away that lasted the ensuing two hours on the Chief dump, was interrupted between shifts by an armistice, and then continued for two hours at Eureka. Each fighter was all in and claimed his opponent had won. A few years after that Conelly was looking for a job and nearly collapsed when Burt put him on the Sullivan pay roll for important installations of large compressors. This was typical of Burt, to admire his adversaries when they had proved themselves.

About 1932, when the New Deal came into existence, Burt acquired the *Salt Lake Mining Review*. As an extreme advocate of individuality and self-reliance, he became one of the harshest critics of the New Deal and of all those connected with it who sought political prestige by the spending of Government funds for the aggrandizement of their own and of the Government's power, to the detriment of private ownerships and controls. However one may look at his extreme views on this subject, as fully expressed in his editorials, one cannot refuse to admire his courage in expressing views that might work to his disadvantage in the conduct of his business. Personally, I always looked forward to his editorials to see what was coming next.

At the recent meeting of the American Mining Congress at Las Vegas, Burt was chosen to tell the truth about uranium as conducted by various individuals in that industry. He did a splendid job of that, just a few days before his death.

In his private mining practice Burt was always ready to take up the cudgels for integrity and high ethical standards and to condemn any concession to lowering of these standards. In his home life he had a great love of his garden and flowers and was worshiped by his family. Burt would give freely of anything he possessed to a friend, and I know of many instances in which he did so.

And so passes a gentleman of the old school and a great soul, from the field of mining. His many friends will miss him and they have tried to comfort his family by their expressions of sympathy.

R. Hartley Sherwood (Member 1921) died Dec. 2, 1955, of a heart attack. Mr. Sherwood was chairman of the board, Stonefort Corp., Indianapolis. He was born in Bristol, Pa., in 1876. Mr. Sherwood graduated from Brooklyn Polytechnic Institute in 1897 and received an M.E. from Cornell University. He began his career in the coal industry in 1913 when he started coal strip mining operations with headquarters in Danville, Ill. Mr. Sherwood founded the Central Indiana Coal Co. in 1917. He was president of the Sherwood-Templeton Coal Co. and one of the founders of Bituminous Coal Research Inc. Mr. Sherwood also organized the Little John Coal Co.

Necrology

Date Elected	Name	Date of Death
1934	Edward W. Ames	Dec. 8, 1955
1953	John F. Barnes	June 9, 1955
1955	Cecil H. Brehaut	Jan. 14, 1956
1920	George L. Danforth, Jr.	Dec. 17, 1955
1945	Ernest Russell Dickie	Dec. 14, 1955
1915	Robert E. Dye	Nov. 18, 1955
1917	V. M. Frey	Jan. 4, 1956
1916	R. S. Handy	Dec. 24, 1955
1907	Joseph Harold Hedges	Jan. 12, 1956
1934	R. E. Heithecker	Nov. 20, 1955
1900	Robert A. Kinzie	Oct. 22, 1955
1921	E. W. Lyon	Feb. 11, 1955
1888	Ellison C. Means	Jan. 20, 1956
1937	James A. Moore	Nov. 30, 1955
1911	C. E. Nighman	Jan. 11, 1956
1946	George A. Parker	Oct. 26, 1955
1951	Harold V. Trask	Dec. 23, 1955
1935	O. W. Wells	Jan. 19, 1956
1944	A. B. Woodward, Jr.	Nov. 13, 1955

MEMBERSHIP

Proposed for Membership
Mining Branch, AIME

Total AIME membership on Dec. 31, 1955 was 23,723, in addition 2825 Student Associates were enrolled.

ADMISSIONS COMMITTEE

R. B. Caples, Chairman; F. A. Ayer, Vice-Chairman; A. C. Brinker, R. H. Dickson, C. R. Dodson, R. B. Fulton, T. D. Jones, F. W. Hanson, Sidney Rolle, F. T. Sisco, O. B. J. Fraser, F. W. McQuiston, Jr., A. R. Lytle, L. P. Warriner.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

Members

Bertrand M. Andreas, Hibbing, Minn.
Malcolm R. Atonell, Duluth
William H. Bennetts, Virginia, Minn.
Stephen B. Bond, Quebec
Harry M. Brindle, Kimberly, Nev.
Robert R. Brooks, Denver.
Herbert K. Conn, Asbestos, Quebec
John R. Ellis, Casper, Wyo.
Tony Enrico, Morenci, Ariz.
Herbert C. Ferkin, Ogden, Utah
George L. Fraser, Magna, Utah
Rueben G. Goetz, Retsof, N. Y.
Evan E. Hall, San Andreas, Calif.
John E. House, Duluth
Martin D. Hughes, Duncan, Ariz.
Joseph M. Johnson, Eveleth, Minn.
Richard H. Little, South Strafford, Vt.
Douglas R. Mabey, Salt Lake City
William H. McCauley, Alhambra, Calif.
Samuel A. Miller, North Olmstead, Ohio
Robert A. Rice, Osburn, Idaho
Julius A. Strande, Hibbing, Minn.
Norman E. Sparks, Carlsbad, N. M.
Verne J. Tarne, Moab, Utah
John J. Thiessen, Fernandina Beach, Fla.
Justo A. Vigil, Lima, Peru

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John V. Andrews, Cleveland
James A. Buchanan, Cheyenne, Wyo.
George S. W. Cumpston, Lima, Peru
Ralph L. Dickey, Cleveland
Elwood Eggers, Salt Lake City
Donald Gasper, Pittsburgh
Robert J. Grisley, Salt Lake City
Wesley R. Harkins, Duluth
Paul M. Kavanagh, Skouriotissa, Cyprus
James F. Powell, Columbia, Tenn.
William H. Price, Salt Lake City
Henry E. Reed, Jr., Williston, N. D.
George I. Reynolds, Jr., Pittsburgh
George H. Rounthwaite, Las Vegas, Nev.
W. W. Schultz, Washington, D. C.
Rudolph A. Warren, Virginia, Minn.
Ray G. Wolfe, Anchorage, Alaska

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Malcolm A. Griffiths, Ghatsila, Chotanagpur, Bihar, India
Paul, E. Hammond, Los Angeles
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George P. McNair, Jr., Morenci, Ariz.
Eugene L. Palusky, Grand Rapids, Minn.
William D. Stark, Logan, W. Va.
Philip J. Urso, Eagle Mt., Calif.
Roger E. Wakefield, Columbus, Ohio

CHANGE OF STATUS

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Maynard F. Ayler, Golden, Colo.
Edward H. Brinley, Potrerillos, Chile
John A. Cronin, Inspiration, Ariz.
Edwin S. Godsey, El Paso, Texas
Dexter C. Hatch, Tahawus, N. Y.
Kenneth G. Hatfield, Albuquerque, N. M.
Charles E. Johnston, Albuquerque, N. M.
Dan E. Lewis, Jr., Manila, P. I.
Douglas F. Lillie, Bancroft, Ont.
George F. McKereghan, Ahmeek, Mich.
David D. McNaughton, Santa Rita, N. M.
Alex E. Nielson, Tempiute, Nev.
Billy W. Nichols, Santa Rita, N. M.
Douglas R. Peck, Dresden, Tenn.
Robert T. Sanden, Hibbing, Minn.
Robert J. Searls, Jerome, Ariz.
Koehler S. Stout, Montana Schools of Mines, Butte, Mont.

Junior to Member

Wayne J. Ahl, Ishpeming, Mich.
Bruce R. Pickering, Tabora, Tanganyika, East Africa

REINSTATEMENTS

Associates

Byron G. Coan, Salt Lake City

Junior

Richard M. McAlexander, Grants, N. M.

CHANGE OF STATUS

Associate to Member

Carlos H. Plenge, Lima, Peru
Leonx H. Rand, Columbus, Ohio

Junior to Member

Arthur C. Erickson, Eveleth, Minn.
Robert C. Horton, Reno, Nev.
Donato L. Russell, Central City, Colo.
Raymond W. Schatz, Columbus, Ohio
Robert E. West, Ajo, Ariz.
Howard F. Ameen, Ishpeming, Mich.

Student to Member

Arnold D. Buzzalini, Staten Island, N. Y.
Julius H. Moor, Palmer, Alaska
John E. O'Neill, Ajo, Ariz.
Bruce W. Stunkard, Aurora, Minn.

Student to Associate

John G. Barnes, Spokane, Wash.

Student to Junior

Hugh M. Miller, Flushing, L. I., N. Y.
Warren B. Schipper, Ely, Nev.

Alvin A. Smith (Member 1948) died Nov. 26, 1955 at Montrose, Colo., Memorial Hospital of a cerebral hemorrhage. He was superintendent of mining operations for Four Corners Uranium Corp., with widespread activities in Colorado, Utah, and New Mexico. Mr. Smith was stricken while in his car at work. He was also general superintendent of Silver Bell Mines Co., Ophir, Colo., where for the past nine years he had made a record for efficient operation and a safety record without a single fatality. Mr. Smith was born in Steamboat Springs, Colo., Feb. 16, 1905.

Charles F. Allen

An Appreciation by

S. A. Falconer

Charles F. Allen (Member 1934), metallurgical engineer, well known in mining circles in this country and South Africa, died suddenly on Sept. 8, 1955 while vacationing with his family at Loveland, Colo.

Mr. Allen was born in Waldenburg, Ark., on Nov. 16, 1911 and spent his early years in the western states. He attended the Colorado School of Mines, graduating with the Bachelor degree of metallurgical engineer in 1934. Prior to World War II he was employed at Nevada Consolidated Copper Co.'s Hurley, N. M., flotation plant and later at Standard Silver & Lead Co.'s Jay Gould mine. During World War II he served with the U. S. Army Corp. of Engineers. Following his discharge from the army, he became associated with American Cyanamid Co. and retained his connection with that company until the time of his death. During these years he served as a metallurgical testing engineer at the Stamford, Conn., laboratory, and later as a technical sales representative in the northwestern states. Subsequently, he was transferred to Africa, with headquarters in Johannesburg, acting as technical representative for South African Cyanamid Pty. Ltd. Later, he returned to the U. S. and for a time was stationed at the Stamford Laboratory, conducting special metallurgical investigations. In 1953 he was appointed project director of the Winchester Laboratories, which at that time were being operated by American Cyanamid Co. for the Atomic Energy Commission. When this contract terminated he was appointed operations manager of Cyanamid's Waterbury, Conn., pilot plant—his last assignment.

Those of us who were privileged to know Chuck intimately will long remember with affection his good humor, his optimism, and ready wit, as well as his capabilities and capacity for hard work. His untimely death marks finis to an unusually promising career in the fields of mineral beneficiation and metallurgy in which he excelled.

Coming Events

Mar. 21-23, American Power Conference, Sherman Hotel, Chicago.

Mar. 23, ESWP-AIME, Pittsburgh Section, joint meeting, Hotel William Penn, Pittsburgh. J. F. Core and C. W. Rountree, Jr., will speak on "Industrial Engineering as Applied to Coal Mining."

Mar. 28-31, AIME, Nevada Section, Reno.

Mar. 28-31, Cordilleran Section, Geological Society of America, convention, Reno, Nev.

Apr. 5, AIME Utah Uranium Subsection, Arches Cafe, Moab, Utah.

April 6, Pennsylvania Anthracite Section, Necho Allen Hotel, Pottsville, Pa. John Cipyak will speak on "Machine Accounting as Applied to Mining Industry."

Apr. 9-11, AIME, Open Hearth and Blast Furnace Conferences, Netherland Plaza Hotel, Cincinnati.

Apr. 9-11, Canadian Institute of Mining and Metallurgy, annual meeting, Chateau Frontenac, Quebec City.

Apr. 18-22, Assn. of American State Geologists, annual meeting, Kenlake Hotel, Kentucky Lake State Park.

Apr. 23-25, Symposium on Rock Mechanics, Colorado School of Mines, Golden, Colo.

Apr. 23-25, American Zinc Institute-Lead Industries Assn., annual meeting, Statler Hotel, St. Louis.

Apr. 23-26, American Assn. of Petroleum Geologists, Conrad Hilton Hotel, Chicago.

Apr. 26-28, Assn. for Advancement of Engineering, seventh annual conference, Virginia Polytechnic Institute, Blacksburg, Va.

May 3-5, AIME, Pacific Northwest Regional Metals and Minerals Conference, Olympic Hotel, Seattle.

May 7-9, American Mining Congress, Coal Convention, Netherland Plaza Hotel, Cincinnati.

May 17-18, Rocky Mountain Petroleum Sections, joint meeting, Gladstone Hotel, Casper, Wyo.

June 16-23, Fifth World Power Conference, Vienna.

June 17-22, American Society for Testing Materials, annual meeting, Hotel Chalfonte-Haddon Hall, Atlantic City, N. J.

Aug. 19-24, Sixth International Symposium on Combustion, Yale University, New Haven, Conn.

Aug. 23-25, National Council of State Boards of Engineering Examiners, Hotel Statler, Los Angeles.

Sept. 4-11, 20th International Geological Congress, Mexico City.

Sept. 26-28, AIME Rocky Mountain Minerals Conference, Salt Lake City. Technical papers will represent all branches, including the Petroleum Div.

Sep. 30-Oct. 1, Society of Exploration Geophysicists, 26th annual meeting, New Orleans.

Oct. 1-4, American Mining Congress, Mining Show, Shrine Auditorium, Los Angeles.

Oct. 8-10, AIME, Institute of Metals Div., Allerton Hotel, Cleveland.

Oct. 8-12, Union of Pan American Assns. of Engineers, fourth convention, Mexico City, Mexico.

Oct. 14-17, AIME, Petroleum Branch, Biltmore Hotel, Los Angeles.

Oct. 22-23, ASA, 38th annual meeting, Hotel Roosevelt, New York.

Oct. 30-Nov. 1, Society of Exploration Geophysicists, annual meeting, New Orleans.

Nov. 1-3, Geological Society of America, Minneapolis.

Feb. 24-28, 1957, AIME Annual Meeting. Roosevelt and Jung hotels, New Orleans.

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